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EXPLOSION ISSUING FROM PORTAL, TEST 15.

Bulletin 56

**DEPARTMENT OF THE INTERIOR
BUREAU OF MINES**

JOSEPH A. HOLMES, DIRECTOR

**FIRST SERIES
OF
COAL-DUST EXPLOSION TESTS
IN THE
EXPERIMENTAL MINE**

BY

GEORGE S. RICE, L. M. JONES, J. K. CLEMENT

AND

W. L. EGY



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FIRST SERIES OF COAL-DUST EXPLOSION TESTS IN THE EXPERIMENTAL MINE.

By GEORGE S. RICE, L. M. JONES, J. K. CLEMENT, and W. L. EGY.

INTRODUCTION.

By GEORGE S. RICE.

This report has been prepared, not only for the purpose of recording the results of the first series of coal-dust tests conducted in the experimental mine of the Bureau of Mines, but also to place before the mining public a description of the mine and an account of the objects sought in its establishment.

Soon after the organization of the technologic branch of the United States Geological Survey, J. A. Holmes, then chief of the branch, concluded that for the solution of various problems relating to the causes and prevention of mine explosions it would be best to carry on large-scale tests in a mine rather than in a surface gallery.

The idea of utilizing the underground passage of a mine for making explosion experiments did not at first meet with general approval among investigators in this country or abroad. Many thought a mine passage was not a proper place in which scientific tests could be conducted, as great difficulty in controlling conditions was expected. However, after all arguments for and against such an investigation had been carefully considered, plans for opening an experimental mine were definitely adopted by the Bureau of Mines in 1910, and an allotment of funds was made to carry out the project. The first endeavor was to find an existing mine suitable for the purpose, but the mines offered by owners were either worked out, were wet, had poor roofs, or presented complications of various kinds, so that it was finally decided to open a mine if a site could be obtained having natural conditions that would fill the various requirements.

A most desirable location was found on property of the Pittsburgh Coal Co., near Bruceton, Pa. The president of this company, Mr. W. K. Field, the general manager, Mr. G. W. Schluederberg, and the chief engineer, Mr. E. J. Taylor, rendered every possible service in

making arrangements for the lease of the property and subsequently in the opening of the mine.

Immediately on completion of the development of the mine to the extent considered desirable for experimental work, the first series of tests was begun.

The number of experiments embraced in the first series, here reported, was comparatively small. The tests chiefly served to try out the mine and apparatus; apart from this end, there was valuable educational service accomplished by certain tests before large audiences of mining men. The fact that coal dust in air containing no inflammable gas may explode is now almost universally conceded in this country. This was not the case prior to the first public test of October 30, 1911, when many still doubted even after having witnessed explosion tests in the Pittsburgh surface testing gallery. However, as a result of the mine test mentioned, those who had previously doubted expressed themselves as convinced.

This explosion test seriously damaged the mine equipment, necessitating considerable delay before repairs could be completed. Then followed further experiments, which were limited in number by lack of funds.

PHENOMENA OF A DUST EXPLOSION.

When a source of heat, for example, an incandescent platinum wire, an electric arc, or the flame from a blown-out shot, is introduced into a cloud of coal dust, combustion takes place between the particles of dust and the oxygen of the air. If the quantity of heat transferred to the dust cloud is relatively small, the combustion may proceed only in the immediate vicinity of the heating agent. If, on the other hand, sufficient heat is available, the temperature of the dust cloud is raised to such an extent that combustion takes place rapidly, developing more heat than can be conducted away. Consequently the adjacent part of the dust cloud is heated to a temperature sufficient for rapid combustion, and by its combustion develops heat, which in turn raises the temperature of the next dust layer. This process continues, and the flame travels through the dust cloud with rising temperature and increasing velocity.

In this report the term "ignition" is used to describe the propagation of the flame beyond the immediate influence of the igniting agent.

When the conditions are favorable, the flame is propagated through the dust cloud with rapidly increasing velocity and rising pressure. A true dust explosion then results, and the velocity of the flame rises rapidly to over 2,000 feet per second and great pressures are produced.

When an explosive gas mixture is ignited at the closed end of a tube that is open at the opposite end, the flame travels with rapidly increasing velocity toward the open end. In the early stage of the gas explosion the flame is usually propagated in the way just described for a dust explosion; that is, the heat evolved by the combustion in one layer of gas raises the temperature of the adjoining layer above the temperature at which ignition takes place. In certain gas mixtures, for example, a mixture of hydrogen and oxygen, after the flame has traveled a certain distance from the point of ignition an "explosion wave," or so-called "detonating wave," is set up, and from this point the explosion is propagated almost instantaneously.

The term "detonation wave" has been defined by Prof. Harold B. Dixon^a as follows:

The expression "*l'onde explosive*" coined by Berthelot, and its English equivalent, "the explosion wave," signify that flame which passes through a uniform gaseous mixture with a permanent maximum velocity. The rate of the "explosion wave" is a definite physical constant for each mixture; the explosion wave travels with the velocity of sound in the burning gas, which itself is moving rapidly forward en masse in the same direction, so the explosion wave is propagated far more quickly than sound travels in the unburned gas. * * * In the explosion wave, each layer of gas is compressed so suddenly that it is raised beyond its ignition point by the heat of compression. * * * I use the word "detonation" to express the burning taking place in the explosion wave, since a "detonator" when struck burns in this way itself and sets up an explosion wave in explosive gases around it.

As the phenomena of coal-dust explosions are in many respects similar to those of gas explosions, it is not impossible that an "explosion wave" or "detonation wave" may be produced by a dust explosion in a tube or gallery of uniform cross section under certain conditions. A mine heading or entry never presents the uniformity of a tube or an artificial gallery, because in a mine passage there are irregularities due to projections of roof and ribs, to turns, to side openings, to timbering, and to great variation in the amount and quality of dust present. Therefore it does not seem probable that in a mine explosion a detonation wave of permanent maximum velocity can occur, although in some localities the disruptive force of an explosion may be so great that in popular language it might be termed "detonation."

When a coal-dust explosion is caused by a blown-out shot, a pressure wave is started that travels at a rate of a sound wave;^b this wave, which in this bulletin has been termed the "shock wave," gradually becomes less and less in amplitude as it travels from the origin. Near the originating blast, if several pounds of black powder

^a Report of Committee on British Coal Dust Experiments, Record of first series, 1910, p. 150.

^b Nearly 1,100 feet per second in a mine passage, varying slightly according to the temperature of the air and the velocity of the ventilating current.

or dynamite has been used, the concussion may be strong enough to raise near-by coal dust into suspension and thus expose the dust to ignition from the flame; but the blast concussion alone can probably not raise dust sufficient for propagation of the flame under the conditions present in the experimental mine for more than about 100 feet. Following the shock or shot wave are other pressure waves which are started by the expanding gases from the burning coal dust thrown into the air by the concussion from the shot. The fact must be emphasized that unless the dust is raised into the air in a cloud it can not explode. If the dust cloud is dense and other conditions are favorable for intense combustion there is a correspondingly rapid production of hot expansive gases with resultant high pressure, causing the pressure waves and the flame to attain high velocities.

The pressure waves started by the explosion are transmitted to the air ahead, and doubtless travel through it at the rate of sound waves. The body of air is forced ahead of the explosion, and as the pressure waves travel through this, the total velocity of any one of the almost infinite number of successive pressure waves with reference to a fixed point is therefore greater than that of a sound wave. When the combustion is relatively slow the advance air waves may get considerably ahead of the combustion or exploding zone. On the other hand, if the conditions for propagation are favorable, the distance that the air waves are in advance will lessen proportionately.

The existence of the advance air-pressure waves is an all-important factor in the propagation of a dust explosion, for if there were none, the coal dust would not be brought into suspension in the air and thus furnish fuel for continuance of the combustion; the English have popularly termed the advance pressure wave the "pioneering wave" and the dust cloud "the pioneering cloud."

There is no doubt that a dust explosion will die away, if a dustless zone is reached, just as soon as the coal dust carried forward in suspension and its unburned gases have been consumed. Similarly, coal-dust explosions can be prevented from starting by so wetting the dust that it can not be raised into the air by concussion. A dustless condition would be equally good, but can hardly be attained in the average mine.

The French (Liévin) experiments indicate that the maximum pressure is exerted in the zone of combustion. This fact is determined from the automatic photographing of the flame itself on the same revolving film as that on which the pressure curve is photographed. The pressure curve is obtained through the agency of a ray of light reflected from a mirror attached to the back of a dia-

phragm which is exposed to and is deflected by the pressure. The pressure curve indicates that the greatest pressure of the explosion is exerted in the zone of combustion. This maximum pressure wave may be termed the "main pressure wave." The records of the European experiments have shown that as the main pressure wave travels ahead it throws off reflex pressure waves that travel back toward the origin, and that if the origin is at the closed end of the gallery, the pressure may be raised above what it was at the time of ignition.

Following the passage of the pressure waves there is a depression below atmospheric pressure, which is caused by the cooling of the gaseous products of the explosion and by the ejection of the gases by the violence of the explosion. The depression causes a violent movement or inrush of air from other parts of the mine, or from the outside, to fill the partial vacuum.

The pressure waves and their movement are very complex even in a single gallery closed at one end, but become more complicated in a mine with two or more passageways. Taffanel, in reports ^a on his third and fourth series of coal-dust experiments conducted at Liévin in a single main passageway 300 meters (1,000 feet) long, has contributed most important information relative to waves developed in a gallery by simple explosions of dust and the arresting of the explosions in various ways.

OBJECTS OF DUST-EXPLOSION INVESTIGATIONS.

A thorough knowledge of the nature of coal-dust explosions, and of the accompanying phenomena, is necessary in order to devise means for the prevention of such explosions, and for the arrest, or rather, the stoppage of explosions if by mischance explosions start. It is therefore necessary to know how coal dust ignites and the various means by which it may be ignited, or inflamed, the chemical processes that take place, the circumstances under which an inflammation becomes a true explosion, and whether detonation sometimes takes place.

In endeavoring to discover the natural laws underlying ignitions and explosions, the bureau considers the factors or variables mentioned below. Some of these are properly subjects for laboratory investigations, and the others for investigations at the experimental mine. The laboratory investigations of several, in particular those relating to the relative inflammability of the different dusts, have been pursued in the laboratories of the bureau at Pittsburgh and in other laboratories, especially in England and France.

^a Taffanel, J., Troisième série d'essais sur les inflammations de poussières; production des coups de poussières, April, 1910; Quatrième série; théorie des explosions, August, 1911.

INFLAMMABILITY FACTORS PROPERLY INCLUDED IN LABORATORY STUDIES.

Factors affecting inflammability that are properly included in laboratory investigations are as follows:

- (1) The chemical composition of typical coal dusts.
- (2) The relative inflammability of dusts containing different percentages of volatile matter, ash, sulphur, and moisture, as tested by different laboratory methods and apparatus and as influenced by:
 - (a) Character of ignition coil, induction spark, electric arc, or flame.
 - (b) Size of coal-dust particles.
 - (c) Method of putting coal dust into suspension.
 - (d) Method of mixture with inert dusts and chemically unstable compounds to give extinguishing effects.
 - (e) Presence of methane and other gases.
 - (f) Size and shape of explosion chamber.
- (3) The physical nature, composition, and affinity for oxygen of each of the component substances that make up the coals from which inflammable dusts are derived (microscopic and chemical investigation).
- (4) The nature and composition of gases distilled from the coal dusts at different temperatures and during different periods of exposure to heat.

FACTORS PARTLY UNDER CONTROL IN EXPERIMENTAL EXPLOSIONS IN THE MINE.

Factors more or less under control in experimental dust explosions in the mine are enumerated below. The bureau is investigating the effect of each in connection with its study of dust explosions.

- (1) Source, composition, and relative inflammability of dust.
- (2) Size and character of dust particles.
- (3) Admixture of inflammable dust with inert dusts.
- (4) Quantity of dust per linear or cubic foot of space.
- (5) Method of loading dust—on shelves, on floor, etc.
- (6) Moisture adhering to dust and not part of its normal composition.
- (7) Moisture on walls, roof, or floor, and not in contact with dust.
- (8) Humidity and temperature of air.
- (9) Air movement in passageways—velocity and direction with reference to proposed explosion path.
- (10) Character of walls, roof, and timbering.
- (11) Turns and curves in path of proposed explosion.
- (12) Proposed branching of the explosion into other entries by prearranged loading.
- (13) Coming together of two explosions by prearranged loading.

- (14) Local widening of explosion passage.
- (15) Number of side openings, such as cut-throughs, not dust laden.
- (16) Dustless zones of various lengths in path of explosion.
- (17) Watered zones of various lengths in path of explosion.
- (18) Stone-dust zones of various lengths and kinds in path of explosion.
- (19) Means of ignition of coal dust, such as:
 - (a) Blown-out shots of various intensities with various explosives.
 - (b) Explosion of fire damp.
 - (c) Electric arcs or glowing wires or blowing out of fuse.
 - (d) Open flame when dust is already in suspension in air.
- (20) Length of mine passage used in test.
- (21) Character of mine passage—double or single or closed at one end, that is, “in the solid.”

VARIABLE EXPLOSION CHARACTERISTICS FOR DETERMINATION IN EACH TEST.

Variable explosion characteristics subject to determination in each test are as follows:

- (1) Velocity of flame at different stages of explosion.
- (2) Velocity of pressure waves at different stages of explosion.
- (3) Relative position of flame and of pressure waves.
- (4) Rise and fall of pressure, above and below the atmospheric, at different points in the course of an explosion.
- (5) Composition of gases at various stages of an explosion, the samples being gathered automatically.
- (6) Temperature of explosion at different stages, as determined by thermocouples, or by strips of metal of different melting points.
- (7) Composition of coke and ash residues from explosion, and their position with reference to direction of explosion.
- (8) Manifestations of violence.

PROBLEMS TO BE INVESTIGATED AFTER DETERMINATION OF STATED VARIABLES.

Below are enumerated problems that the bureau will investigate after having determined satisfactorily the influence of the variables mentioned above:

- (1) Length of dust-laden zone (of given cross section) measured from a blown-out shot (of certain strength) that, through the ignition of its dust, creates sufficient pressure waves to raise dust beyond, and thus permit propagation of the explosion beyond the influence of the blown-out shot.
- (2) Length of a dust cloud, which, when ignited by an arc or open flame, will cause pressure waves to stir up the dust beyond and cause a propagation of the ignition or inflammation.

(3) The circumstances under which the ignition or inflammation becomes a strong explosion, productive of violence.

(4) Possibility of producing a coal-dust "explosion wave" or "detonating wave" of permanent maximum velocity.

(5) The presence, origin, and character of reflex waves thrown off from the main explosion pressure waves and traveling back toward the point of ignition.

(6) Chemical reactions taking place at different stages of an explosion.

OUTLOOK FOR SOLUTION OF COAL-DUST PROBLEMS.

The solution of all the problems connected with coal-dust explosions, as outlined above, will not be simple, but with the systematic prosecution of the investigations it is probable that the determinations will be sufficiently complete to enable satisfactory planning of adequate means of preventing explosions and the development of valuable secondary safeguards.

Meantime testing stations in France, England, Belgium, Germany, and Austria are prosecuting similar investigations, and through friendly interchange of information it is expected that the advance will be more rapid than would be made by the bureau's investigators alone.

OUTLINE OF VARIOUS TESTS MADE IN THE EXPERIMENTAL MINE.

In addition to the explosion experiments described in this report, a considerable number of tests was conducted in the exterior gallery to determine the conditions under which ignition of coal dust by an electric arc could be obtained. Subsequently, while the mine headings were being extended, tests were made to determine the relative efficiency in blasting coal of different kinds of permissible explosives, also of black powder, under the natural conditions offered by the mine. With a view to determining the vitiation of mine atmosphere by the use of these explosives, samples of their gaseous products were obtained and analyzed. Some experiments with the treatment of coal dust by calcium chloride for prevention of ignition were made. An important series of tests of two types of gasoline locomotives was conducted with a view to testing mechanical control, and to determining to what extent the exhaust vitiated the mine atmosphere under different conditions of use and of engine adjustment. To repeat, then, although only a small number of large coal-dust explosion tests was made, the mine was constantly used for experiments.

In making explosion tests, one of the most important advantages connected with the experimental mine as compared with a surface gallery is the possibility of experimenting on coal-dust ignition when

small quantities of methane are present without the large wreckage that would be liable to result in a surface gallery. The study of the effect of the presence of small quantities of methane is of great importance; probably more often than has been suspected, methane has been present in an amount too small to be detected by a safety lamp yet has been influential in the first ignition and propagation of a dust explosion.

Among other possibilities offered by the experimental mine when equipment suitable for each class of experiments has been installed is that of testing the relative dust production of different kinds of undercutting machines, and of investigating the safety features of underground electrical installations, including tests of devices for greater safety in the use of the electric-trolley locomotive, and of studying problems of mine ventilation, such as the efficiency of fans, doors, and stoppings, and the determination of the frictional resistance to the passage of air currents offered by the ribs, roof, and timbering of mine entries and rooms. The figures currently used as coefficients of friction of air passing through underground passageways are approximations, and the wide range of values of the coefficients given by different authorities shows the need of more positive determinations. Altogether, the possibilities for usefulness of the experimental mine are limited only by the appropriations that may be available from year to year.

The scientific study of coal-dust explosions in the experimental mine has been only begun. The most important results of the preliminary series of experiments described in these pages have been: (1) To convince the mining public that coal-dust explosions can be produced at will in a mine as well as in a gallery, and (2) to show that relatively violent explosions of coal dust can be produced by a violent means of ignition in a comparatively short length of passageway. The second result is of importance because it shows the necessity of preventing an ignition of coal dust rather than of attempting to limit an explosion by zonal treatment except as the latter is made a secondary safeguard.

ACKNOWLEDGMENTS.

For assistance rendered in developing the experimental mine the writer of this introduction, who was placed in general charge of the investigations, desires to express appreciation of the inspiration and guidance received from Director Holmes, and the hearty and efficient cooperation of his associates, L. M. Jones, mining engineer; H. C. Howarth, mine foreman; J. K. Clement, physicist; and W. L. Egy, assistant physicist. Many others connected with the bureau, but not directly connected with the experimental-mine work, contributed valuable assistance, among whom are H. H. Clark, electrical engineer;

J. C. W. Frazer, chemist; J. W. Paul, mining engineer; J. J. Rutledge, mining engineer; H. I. Smith, assistant mining engineer; Clarence Hall, explosives engineer; and O. P. Hood, chief mechanical engineer.

The driving of the mine headings or entries, their lining with concrete, the outside and inside construction of the mine, and the preparation of the mine for each test were under the immediate charge of L. M. Jones, mining engineer, and H. C. Howarth, mine foreman.

The setting up of the instruments, their wiring, their operation, and the interpretation of their records were in charge of J. K. Clement, physicist, and W. L. Egy, assistant physicist.

The analyses of coal, coal dust, shale, and rock dust were made by A. C. Fieldner, chemist; and the analyses of mine atmospheres and mine gases were made by G. A. Burrell, chemist.

DESCRIPTION OF MINE AND EQUIPMENT.

There was no precedent to follow in developing an experimental mine of the magnitude desired. At Segengottes, Austria, in the Rossitz coal-mining district, there is an underground testing gallery, the main passage of which was an abandoned drainage tunnel lined with stone throughout. It is 293.7 meters (964 feet) long, but it is narrow, being only 1.3 to 1.44 meters ($4\frac{1}{4}$ to $4\frac{3}{4}$ feet) wide, and the outer end terminates in a group of houses. Tests of the inflammability of coal dust are carried on in the inner portion of the gallery, but owing to the surrounding conditions large-scale, violent tests are not attempted. Birmingham University, England, has for the instruction of mining students in the use of breathing apparatus and lamps some underground passages in an old mine, but these are not used for explosion tests. Henry Hall, an inspector of mines in Great Britain, experimented with coal dust in 1876 in an adit that was 135 feet long; subsequently, in 1890 to 1893, he conducted series of coal-dust explosion tests in disused shafts about 200 yards in depth.

The many surface galleries erected in the past for experimenting with coal dust are described in Bulletin 20 of the Bureau of Mines.^a

SELECTION OF A MINE.

The requirements considered by the bureau in planning for an experimental mine were:

(1) The mine should be in a coal bed that produced dust of inflammable character and the other conditions of which favored the possibility of dust explosions.

(2) The mine should be naturally dry and preferably self-draining so as to allow experiments with dry coal.

(3) The main openings of the mine should be drifts, so that complications from possible wreckage of a shaft would be avoided, thus keeping down the expense and permitting a quick entrance to the mine after explosions.

(4) The mine should be as free as possible from methane; that is, the mine atmosphere should contain less than 0.1 per cent, in order that experiments might be made in an atmosphere practically free from inflammable gas.

^a The explosibility of coal dust, by G. S. Rice, with chapters by J. C. W. Frazer, Axel Larsen, Frank Hane, and Carl Scholz. 1911. 204 pp. Revision of U. S. Geological Survey Bull. 425.

(5) A natural-gas pipe line should pass near enough to the mine to make feasible the laying of a branch line that would furnish a supply of gas (methane and ethane) for experiments in combination with coal dust and air in varying proportions, or with mixtures of gas and air only.

(6) A good water supply should be available both for boiler use and for fire protection as well as for experiments.

(7) The location should be accessible to a railroad so that the coal produced could be loaded on railroad cars, yet should be so isolated from houses or other buildings that violent explosions would not endanger them.

After much search these requirements were all met by a site near Bruceton, Pa., 13 miles southwest of Pittsburgh, on the Baltimore & Ohio R. R. Here, the well-known Pittsburgh coal bed outcrops along the sides of the valley of Lick Run. In a side ravine, in which there is a small stream, a place in the outcrop was found sufficiently

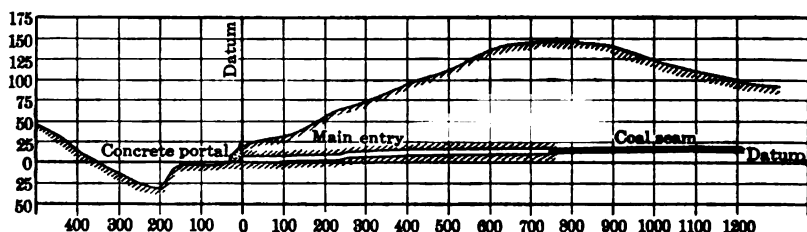


FIGURE 1.—Profile of main entry of experimental mine.

back from the main frontage to allow entrance into the middle of an extensive area of the coal. At the place selected for the drift openings the bed dips gently to the north, and as the opening entries are driven to the southwest they are self-draining. Many samples of mine air have been taken and no appreciable amount of methane has been found; even after the mine has been sealed for 14 hours a maximum of only 0.06 per cent was found at the face of a heading. The drift openings are high enough above the ravine bottom to provide space for a convenient refuse dump and for other features connected with the development of a mine. The hillside above the outcrop rises steeply, so that there is a good cover over the main headings. The crest of the hill is 132 feet vertically above the mine (fig. 1). Across the ravine in front of the mine openings there is a hillside that provides a natural barrier for catching material thrown out by a violent explosion and deflects the air waves upward, a protective feature of great importance in a populated district.

UNDERGROUND DEVELOPMENT.

Drift openings were started in December, 1910, and entries were driven into the coal bed. Drifting proceeded intermittently up to October, 1911, when the first series of coal-dust tests was begun. The underground passages then consisted of two main parallel entries a little more than 700 feet long, each 9 feet wide, with a 41-foot pillar of coal between them. There are crosscuts or "cut-throughs"^a between the entries every 200 feet. A diagonal heading 198 feet long connects the air course with a third opening (fig. 2); it enters the air course at an angle of 55° at a point 117 feet from the mouth of the air course. Fifty-five feet from the mouth of the air course there is a fourth opening, an air shaft that is offset 6 feet from the entry.

This shaft is intended for ventilating purposes only, while the mine is being developed, or as an auxiliary opening in case an experimental explosion should wreck the other openings. The diagonal heading was made in order to provide an opening for ventilation purposes; it was turned off from the main air course, with the expectation that the chief force of an explosion wave would pass directly out of the air course.

The main entries were driven in the coal bed "on the faces," that is, at right angles to the principal vertical cleavage. These entries are of the usual width employed in working the Pittsburgh bed, nominally 9 feet, but ranging between 9 and $9\frac{1}{2}$ feet. It was first intended to place the cut-throughs 100 feet apart, but owing to the expense of constructing tight stoppings strong enough to resist explosions, the distance was changed to 200 feet. In driving the entries, line brattices are carried from the last cut-through up to the face of each entry. Fortunately the natural roof in by the concrete lining is sufficiently strong so that only a few cross timbers have as yet been necessary. Side posts have been put in, not to support the roof but for attachment of the side shelves on which to lay coal dust. In the first two tests these posts were exposed, and hence were blown down; subsequently they were recessed (Pl. II, A) into the coal rib, so as not to be exposed to the violence of the explosion. Ordinary mine tracks of 42-inch gage are laid in each entry for handling coal cars and for use in making tests of gasoline and electric locomotives.

REINFORCED CONCRETE LINING.

The main entry is provided with an arched portal of heavily reinforced concrete (Pl. II, B, and figs. 3 and 4) with retaining wall, wing walls, and buttresses built "en masse;" the walls are carried down to the solid formation, and the buttresses to a limestone $3\frac{1}{2}$ feet below the coal.

^a This term is employed in the bituminous coal mining laws of Pennsylvania of 1911.

The reinforcement consists of $\frac{1}{2}$ -inch, $\frac{3}{4}$ -inch, and 1-inch round rods of mild steel placed vertically, diagonally, and horizontally. The vertical rods are set in drill holes in the limestone. The stresses

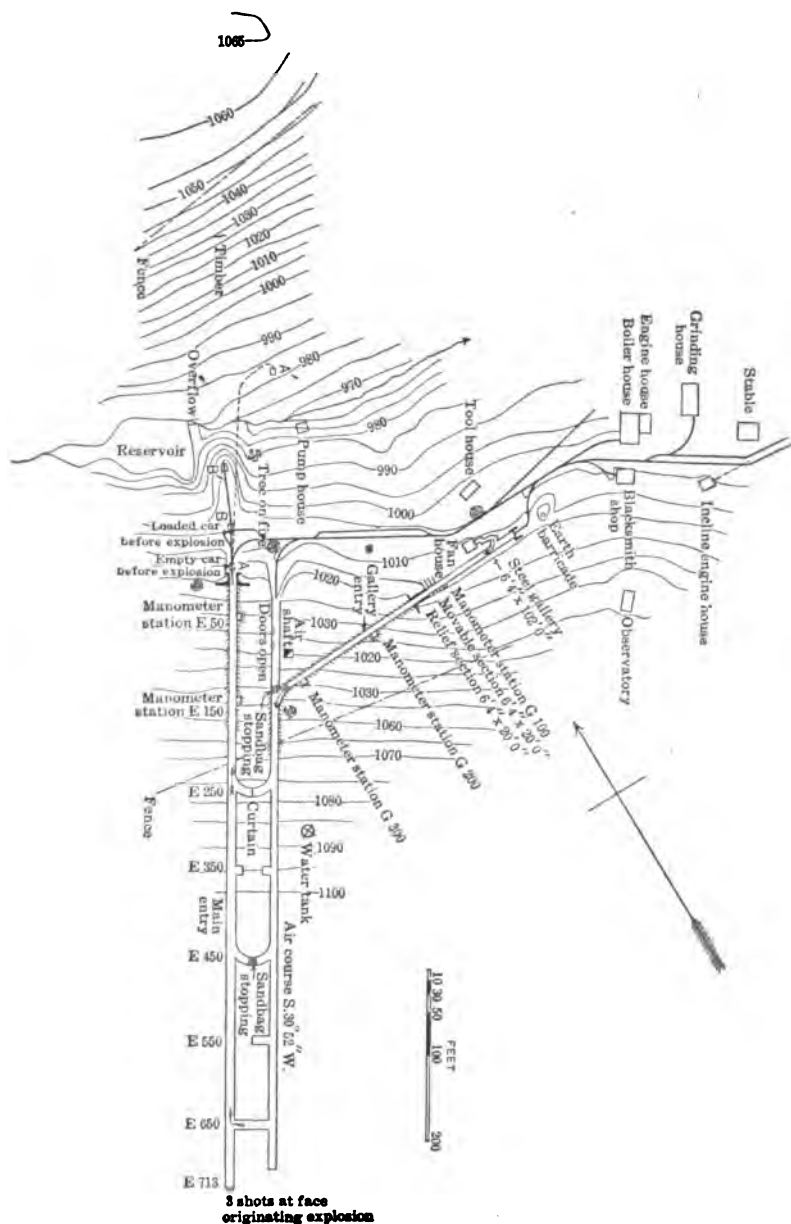


FIGURE 2.—Plan of experimental mine, also surface contours, at time of explosion of October 30, 1911, showing path of mine car thrown by explosion.

considered were (1) those due to forces directed outward from the mine, and (2) a bursting pressure acting at right angles to the axis of the entry or tunnel. The wing walls were also designed as retaining



A. DUST SHELVES IN MAIN ENTRY.



B. FINISHED PORTAL.

walls for the roof shale and dirt cover over the arching of the tunnel. Some of the reinforcement at one stage of construction is shown in Plate III, A. The arrangement at the entrance provides for the future addition of a counterweighted door, sliding vertically in grooves, for use in special experiments requiring a perfectly quiet atmosphere.

It was intended that a similar entrance should be placed at the mouth of the air course. Owing to the lack of funds this was not completed for the tests of the first series, but has since been erected. The air-course entrance was not intended to be used for explosion tests during the first series. A reinforced-concrete portal is provided for the

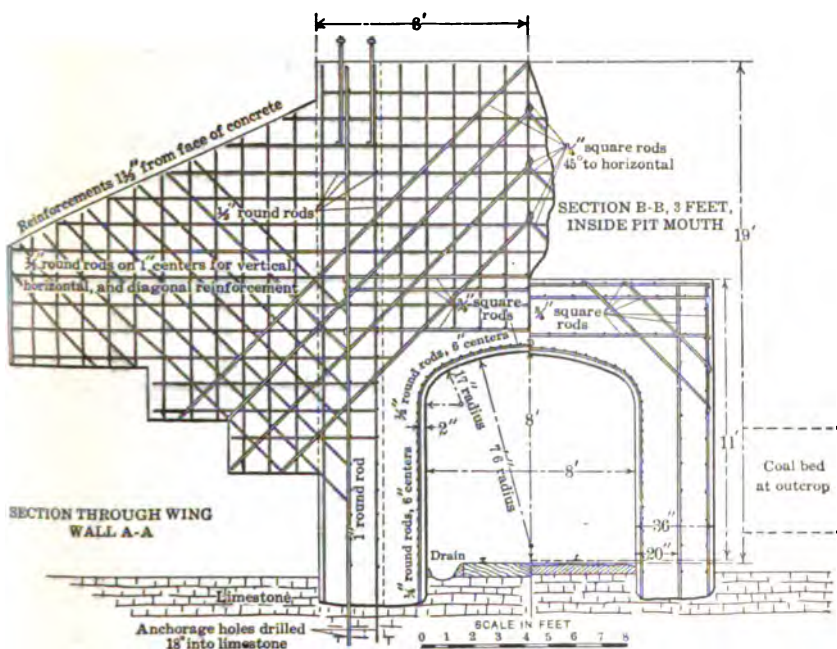


FIGURE 3.—Cross section of reinforced-concrete portal of experimental mine.

diagonal heading similar to that of the main entrance except that the walls are not so heavy, the heading being smaller in cross section.

It was necessary to line the outer parts of the entries on account of the poor roof; indeed this had to be supported by heavy timber while the entries were being driven. It was manifest that timber for lining would not be satisfactory in explosion tests, as the timber would be likely to be blown out. As the best material for lining seemed to be reinforced concrete, a type of lining was adopted similar to that so extensively used in the Béthune mines, Pas de Calais district, France, as designed by J. Lombois, principal engineer.^a The con-

* Lombois, J., Revêtement en béton armé des bouvettes et des bures aux mines de Béthune: Bull. Soc. Ind. Min., ser. 4, vol. 6, 1907, pp. 195-205.



A. REINFORCEMENT OF PORTAL.



B. OBSERVATORY AND OTHER BUILDINGS.



C. EXTERNAL EXPLOSION GALLERY.

inches apart farther in. The arches were made in two halves, each half extending from the foundation to the center of the crown, where they were joined together by a bolt passing through a loop in the end of

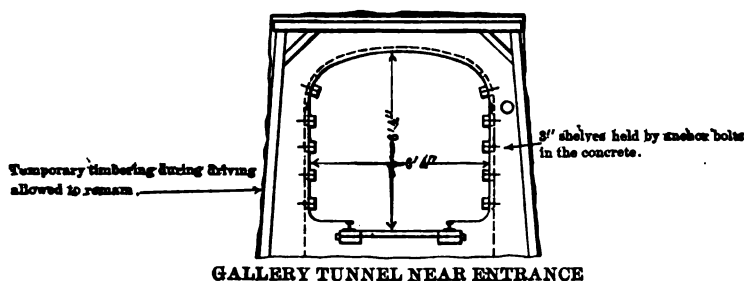
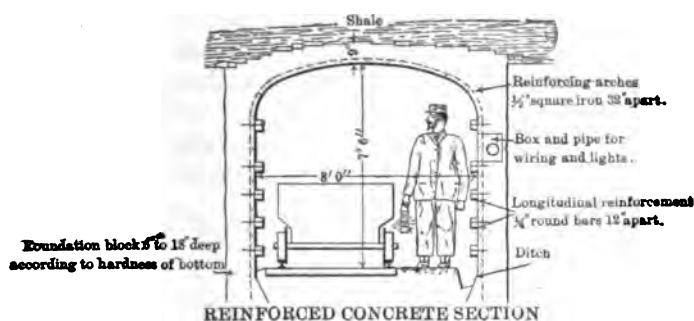
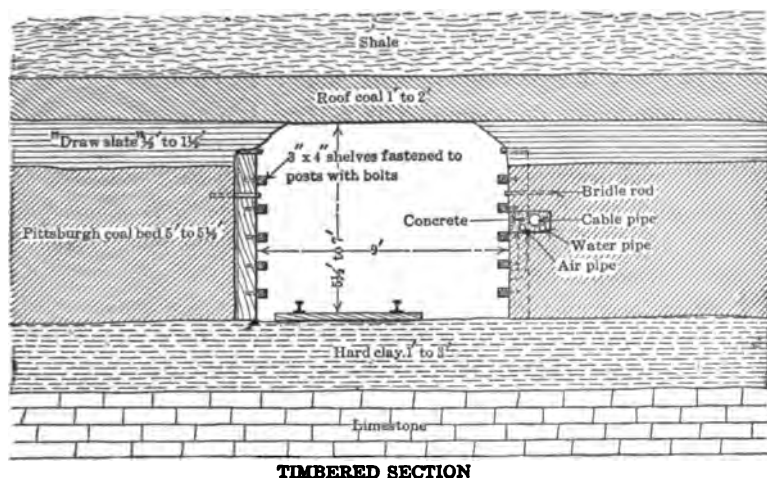


FIGURE 5.—Cross-sectional views of mine gallery.

each half arch. The horizontal reinforcement consists of $\frac{1}{4}$ -inch rods placed 12 inches apart (see fig. 5) and wired to the arch rods at the intersections. The concrete lining was put in place over forms of col-

lapsible type which could be moved from point to point as the work advanced. These forms were similar to those designed by Lombois.^a

While the concreting material of the lining was being placed, bolts were inserted for the support of five shelves on each side to be used in coal-dust explosion experiments. There was also laid behind the wall a 4-inch steel pipe in which were placed electrical cables, a 2-inch pipe for a compressed-air line, and a 2-inch water main with hydrant boxes every 100 feet. The arched lining, which is finished smooth, is 8 feet wide in the clear, and 7 feet 6 inches high from the top of the track ties to the arch; near the entrance the roof has a slight rise and the floor descends, so that the arch is 8 feet high at the portal. The diagonal heading or gallery is smaller, being 6 feet 4 inches wide and 6 feet 4 inches high (fig. 5).

In constructing the lining in the outer part of the main entry and in the diagonal heading or gallery, the preliminary timbering had to be left in place above the arching as the roof was too weak to allow its removal. Adjacent to the portals the arches were constructed in the open; they were afterwards covered with dirt, which was filled in to the top of the retaining walls. During the first series of experiments the concreting was extended into the main entry, making its total length 169 feet; the entire diagonal heading, 198 feet long, was lined, and in the air course there was a lining from a point 20 feet outby the connection with the diagonal heading or gallery to a point 65 feet inby the junction. At the wide space at the junction the roof had been supported with railroad rails which were left in to supplement the strength of the reinforced lining.

THE PITTSBURGH COAL BED.

The Pittsburgh bed, in quality, extent, and uniformity, is one of the most remarkable coal beds in the United States, if not in the world. It extends southwestward from the vicinity of the city of Pittsburgh to the middle of West Virginia, where there are scattered areas in the hilltops, a distance of about 125 miles; on the west it extends into Ohio; the width is 75 to 100 miles, but not all of this area is underlain with the Pittsburgh bed, as in places the coal is cut out in the valleys of the larger streams and on some of the anticlines that separate the several basins. Although varying in different basins, in any one basin the coal is remarkably regular in quality and thickness. In certain areas the bed produces the highest grade

^a Lombois, J., *Revetement en béton armé des bouvettes et des bures aux mines de Béthune*: Bull. Soc. Ind. Min., ser. 4, vol. 6, 1907, pp. 195-205. The reinforcement was placed in the concrete arch to help support the roof. It was thought that the weight of the roof, together with its resistance to shear, could be overcome by the upward pressure of an explosion. This proved not to be true; the arch and roof lifted, causing the arch rods to break through the facing (see Pl. VIII, C). In future construction the arch reinforcement will be placed farther from the face of the concrete, and the arch thickened to resist the force of an explosion.

of coking coal, the coal from the Connellsville district making the standard coke of the country; in other basins or districts the bed produces gas and steam coals. The thickness ranges from 5 to 10 feet, but in any one basin the thickness varies little.

At the experimental mine, which is in the so-called "gas-coal" district, the bed is 5 to 6 feet thick, averaging about $5\frac{1}{2}$ feet. Above the coal there is a layer of soft shale ("draw slate") a few inches to 2 feet in thickness, averaging about $1\frac{1}{2}$ feet. This "draw slate" falls or is pulled down with the pick as soon as the coal has been removed. In some places it has to be brought down by light shots. Above the "draw slate" there is a "top coal" or roof coal 1 to 2 feet thick; it in most places is shaly, and in the district in which the experimental mine is located it is not sufficiently pure to be regularly taken down and used for fuel purposes. This top coal, when not broken by blasting, makes a good roof, except near the outcrop. The main roof above the top coal consists of shales of little strength; in some places, but not in the vicinity of the experimental mine, the Pittsburgh bed is overlain with a strong sandstone. The floor is clay, which is hard when first mined, but subsequently air-slacks. At the mouth of the experimental mine the clay is 2 to 3 feet thick and is underlain with a limestone stratum.

The coal bed proper at the experimental mine is free from continuous partings except for two thin bands of shale about 3 inches apart, a little below the middle of the bed. The upper band is one-eighth to three-fourths of an inch thick and the lower three-fourths to one and one-fourth inches thick. The "faces" or "butts" of the coal (Pl. IV, *A*) are strongly marked throughout the whole Pittsburgh coal bed and are of great importance in mining. The coal itself is fairly hard, but owing to the planes of cleavage it tends to break to cubical pieces (Pl. IV, *B*) in blasting and in subsequent handling. The regular method of mining in the Pittsburgh district consists of undercutting the coal either by hand or with machines, generally the latter, and then blasting down with two shots in entry work, or two to three shots in the rooms. In general, through the gas and steam coal districts about one-third of the coal produced is nut coal or finer, and two-thirds is "lump."

The average proximate analysis of coal from the Pittsburgh bed from face-section samples gathered in the experimental mine is as follows:

Moisture.....	2.7
Volatile matter.....	36.0
Fixed carbon.....	55.0
Ash.....	6.3

 100.0

The average ultimate analysis is as follows:

Hydrogen.....	5.38
Carbon.....	76.16
Nitrogen.....	1.53
Oxygen.....	9.23
Sulphur.....	1.40
Ash.....	6.30
	100.0

SURFACE EQUIPMENT OF THE MINE.

The surface equipment of the experimental mine consists of an external explosion gallery, ventilating fans, a power plant, a coal-dust and rock-dust crushing and grinding plant, an observatory and control station in which recording instruments are placed, an incline, a coal tippie and chute for loading railroad cars, a reservoir, pump house, blacksmith shop, barn, tool house, and a water-tank and hydrant system for fire protection.

EXTERNAL EXPLOSION GALLERY.

The explosion gallery (Pl. III, *C*) consists of a steel tube 6 feet 4 inches in internal diameter and 122 feet long, set in line with the diagonal heading, or "gallery slant," and 20 feet from its mouth. Between the tube and the mine portal there is a U-shaped passage (fig. 6) of heavily reinforced concrete; it has the same diameter (6 feet 4 inches) as the steel tube and the diagonal heading. The roof of this passage is closed by flat plates resting loosely on 2½-inch round tie rods (connecting the tops of the concrete walls) and when necessary is weighted down with sandbags. This loose covering is 20 feet long by 6 feet 4 inches wide and serves as a great relief valve for the protection of the tube and ventilating fan in case of a very violent explosion in the mine.

In order to make the external steel gallery available for small explosion experiments outside the mine itself, the inner 20-foot section, adjacent to the U-shaped passage, can be rolled to one side; then, after the mouth of the diagonal heading has been closed, the steel gallery is isolated from the mine and tests can be conducted in it without interfering with work in the mine. During such periods mine ventilation is carried on by means of a fan set at the top of the air shaft. The steel gallery when thus isolated is 102 feet long and has practically the same dimensions as the gallery for testing explosives at the Pittsburgh station. There is, however, one difference. The outer end of the steel gallery at the mine is left open except for a board or plank stopping which can be slid into place in the concrete framework. This stopping is to provide for relief of violent pressure and therefore is made comparatively thin (of 2 to 4 inch



A. UNDERCUTTING A BREAK-THROUGH.



B. COAL BROUGHT DOWN BY SHOT

Note characteristic cleavage.

plank) in order that it may break and thus lessen the strain on the tube. Twenty-five feet from the outer end of the gallery there is a branch (fig. 2) 6 feet 4 inches in diameter, set at an angle of 60° . This branch is for connection to a fan for use in producing a venti-

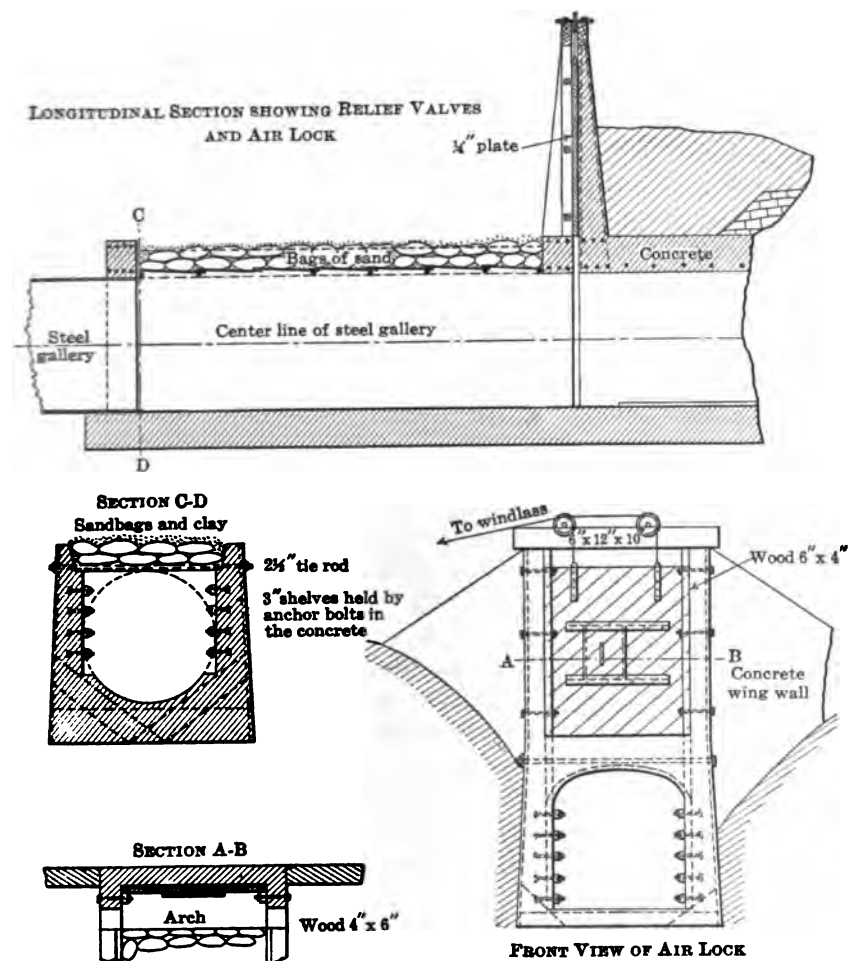


FIGURE 6.—Sections of passage between steel gallery and diagonal entry.

lating current either at the time of an explosion or in the ordinary mine operations.

CANNON FOR IGNITION SHOTS.

For testing explosives or for coal-dust ignition a special cannon similar to those used at the Pittsburgh station is employed. The cannon has a bore hole $2\frac{1}{4}$ inches in diameter and 18 inches deep; it is mounted on a truck which can be wheeled to the outer end of

the steel tube, so that the cannon can point into it through a hole in a heavy plank stopping. The truck has the same gage as the mine tracks, so the cannon can also be used in the mine for making a test in which the equivalent of a blown-out shot is desired. Although long-flame explosives like black powder or dynamite, when used in coal-face shots and so prepared that they will blow out, will readily ignite bituminous coal dust of sufficient purity and quantity, it was found best for experimental purposes to start experimental explosions by means of a blown-out shot from the cannon in order to obtain uniform initial pressures. Another advantage gained by the use of the cannon is that coal-dust explosions can be originated at any predetermined point along an entry or heading of the mine, thus giving conditions similar to those that may occur when a "brushing" shot is fired in an entry or airway.

VENTILATING SYSTEM.

It is intended to have the ventilation at the mine such that it will be possible to duplicate almost any condition liable to occur in an ordinary mine. The entries are double, thus allowing a regular coursing of the air. In future some stub entries will be driven to permit experimenting in a stagnant atmosphere.

The fan, which is now (March, 1912) on hand but not yet installed, is a centrifugal fan of standard make. It is designed for a capacity of 80,000 cubic feet of air per minute at a pressure of 2 inches water gage, or 15,000 cubic feet of air per minute at a pressure of 6 inches water gage. It will be driven by a 100-horsepower steam engine; a belt connection, instead of direct connection, will be used, as the former provides greater flexibility in case of a serious concussion or a sudden stoppage of the fan through violence. The fan is reversible and will be mounted at the end of the branch of the external steel gallery. There will be several relief doors in addition to the protection afforded the fan by its being set to one side of the gallery. Unlike the practice at foreign testing galleries, at which the fan is cut out by valve arrangements a few moments before an explosion is started, it is proposed to run this fan during an explosion in order to parallel conditions that occur in mine disasters.

Unfortunately, sufficient funds were not available for erecting the large fan before the first series of explosion tests, so that a small fan was temporarily set up in place of the large one. The small fan has a capacity of 5,000 to 10,000 cubic feet of air per minute at pressures of 1-inch and $\frac{1}{2}$ -inch water gage and is run by a gas engine. It was originally placed near the top of the air shaft and connected therewith for the ventilation of the mine under ordinary conditions, as in entry driving. It was removed from the air shaft and connected with the

tube during the first series of tests. It will be returned to the air shaft when the larger fan has been erected.

There has been much contention among mining men as to the effect of a ventilating current on the initiation and propagation of a dust explosion. Some claim that a dust explosion always "goes against the air"; that is, advances toward the intake air on account of the additional oxygen it meets in an opposing current of fresh air. They claim further that this condition has most effect in wintertime when the intake air is colder than the return air and hence has greater density and contains more oxygen per cubic foot. Others have thought that the reason that coal-dust explosions are more apt to traverse the intakes than the returns is that the intake air, cold and carrying little moisture, dries the coal dust, whereas the return air is, as a rule, nearly saturated, and the coal dust in the return passageways is apt to be damp. Many have believed that it was a case of the explosion seeking the dry coal dust, whether this is to be found on intake or return. Further, some have suggested that a high velocity of the air current greatly increases both the risk of a dust explosion and its violence. By conducting identical tests except for the direction of the air current, one set of tests with the air moving in one direction, with different velocities, and another set with the air moving in the other direction but with similar velocities, the influence of the direction and the quantity of air current can probably be determined within reasonable limits.

POWER PLANT.

The power plant consists of one 60-horsepower boiler of locomotive type, placed in a galvanized-iron boiler house sufficiently large to permit the erection of a duplicate boiler. There are the necessary steam appliances, hot-water heater, pump, and other fittings. The engine plant is in an annex to the boiler house, and consists of an 8-horsepower steam-generator set, the speed regulation of which is close with change of load.

The generator set is for the purpose of supplying electricity for the recording instruments and for lighting the mine, stations, and buildings. It is expected that in the future a sufficiently large generator set will be installed to enable experiments to be made with electric machinery, electric motors, and electric coal cutters. Since the first series of tests, a single-stage air compressor with a capacity of 174 cubic feet of air per minute at atmospheric pressure and capable of compressing to 100 pounds has been installed for the immediate purpose of running a "puncher" air machine. In the next series of experiments compressed-air jets assisted by the fan will be tried for cleaning the dry dust and soot from the shelves and the walls of the entries.

COAL-DUST AND ROCK-DUST GRINDING PLANT.

For extensive coal-dust experiments such as have been planned, a large quantity of coal dust will be required. For example, if 2,000 feet of entry is to be loaded with coal dust to an extent of 2 pounds per linear foot, 4,000 pounds or more will be necessary. Sufficient coal dust made in the ordinary way can not be easily obtained, nor is it uniform when procured. Therefore the dust is made artificially from the coal mined at the face. In order to obtain comparable results when making coal-dust explosion tests, the coal after having been crushed is uniformly ground until 95 per cent of it will pass through a 100-mesh sieve, as determined by frequent tests of samples. During the first series of experiments the crushing and grinding machinery had not been completed, and the coal dust had to be ground at the Pittsburgh station. For the second series all of the dust will be ground in the mine plant.

The crushing and grinding plant is housed in a building, the first story of which is concrete, the upper part being of galvanized iron on wood framing.

The coal, or the rock if rock dust is desired, is transported in mine cars to the second floor of the building and shoveled on a bar screen with 4-inch spaces, and the larger pieces are broken by sledge until they pass through; a hopper receives the screened material and delivers it to a hammer crusher which breaks it into smaller pieces three-eighths of an inch in diameter and finer. From the crusher the coal is taken by an elevator to a bin that feeds the pulverizer.

The pulverizer is an impact machine. Its essential part is a beater with two blades, which revolves in a steel-lined box at a speed of 2,700 revolutions per minute. The blades break and pulverize the material, and act as fans, the effect of which, assisted by the suction from a large fan above, is to lift the finely ground material through a conical separator provided with deflectors. The air current rising through the separator with a lessening velocity, caused by the enlarging cross section, has a sorting action; the coarser particles drop back and the finer dust is drawn upward to the fan. There are dampers at the top of the separator which can be adjusted to control the air current and cause the coarser dust to fall back to the beaters for finer grinding. On passing through the fan the dust is blown into a large upright pipe which connects above to a "cyclone" collector which gathers all but the lightest dust. The lightest dust and the excess air from the top of the collector are blown into an adjacent collector consisting of a large number of cotton tubes or "stockings," by means of which even the finest dust is strained out as the air discharges through the fabric. The dust gathered by the "cyclone" and the tubular collectors drops through pipes into a tight steel-plate bin.

There is practically no loss of dust. The steel bins are covered in order to lessen the danger from an explosion of dust in the house, and to save the float dust, which, when pure, is probably very dangerous in mines, so that its inclusion with the coarser dust is essential in making tests of the explosibility of coal dust.

On the second floor of the house there is a mine-car track which runs out on the level of the yard sidings. The dust bin is above the track, so that cars can be loaded and taken into the mine with little labor. The crushing and grinding plant will also prepare rock or shale dust to be mixed with coal dust, or placed on shelves, or used in other ways in the tests for preventing or checking coal-dust explosions.

COAL-HANDLING EQUIPMENT.

When entries or rooms are being driven, the loaded coal cars are hauled from the mine by mules or by gasoline motors. The track has a slight down grade from the mine to passing tracks in the vicinity of the boiler house, grinding house, and blacksmith shop, into each of which cars may be run for the easy handling of timber and other material. At the lower end of the yard there is a gas-engine hoist for lowering the mine cars down an incline to the tippie and pulling back the empty cars.

As coal is mined only in extending the headings and in preparing for experiments, the tippie structure is simple, and there are no screens.

WATER SUPPLY AND FIRE PROTECTION.

There is a rivulet in the ravine in front of the mine. An earth dam has been thrown across to make a reservoir. The water is of good quality for boiler supply. A steam pump discharges the water into a 5,000-gallon tank on the hillside above the mine, at an elevation that gives a pressure head suitable for fire fighting—40 pounds per square inch at the mine hydrants.

OBSERVATORY AND CONTROL STATION.

Situated on the hillside, commanding a view of the three entrances to the mine, is the observatory and control station (Pl. III, *B*). It is built of reinforced concrete, the roof being constructed of railroad rails covered with concrete. The front wall, the one facing the mine, is 2 feet thick, and has small windows through which the external evidence of the experiments can be watched in safety. The external recording instruments, the chronographs, time markers, circuit-breaker recorders, and other instruments, as well as the shot-firing connections and firing buttons, are in this building. A detailed discussion of the instruments is given in the section of this

report entitled "Description of Instruments for Measurement of Velocity and Pressure."

UNDERGROUND EQUIPMENT.

It has already been mentioned that tracks are run through the entries to allow easy handling of materials, as well as the transportation of the coal mined at the face. The mine is now equipped with 24 coal cars and one car for concrete work. For coal cutting there is a supply of hand augers, picks, shovels, and other tools. Recently, since the end of the first series of tests, an air compressor has been added, also an air "puncher" machine with which a 5-foot undercut can be made.

A 2-inch water line is installed along the main entry, and another line through the diagonal heading into the air course. These pipe lines are placed in grooves behind the concrete lining, or where there is no lining, in grooves in the coal, the grooves being faced with concrete. The pipes are thus protected to prevent damage during experiments. There are hydrant boxes every 100 feet for attaching hose for fire protection and for washing down or wetting the coal dust.

COAL-DUST SHELVING.

Before making coal-dust explosion experiments it was necessary to plan adequate arrangements for loading the mine headings with coal dust. It is important that the test conditions can be duplicated from time to time. In the surface galleries there is an iron, wood, or concrete floor, which can be swept clean after an explosion test. In the Liévin gallery, in France, the dust is spread on the floor (concreted at the inner end, wood covered at the outer end), and the cleaning is accomplished by increasing the speed of the fan and then raising the dust with a compressed-air jet from a hose, the cleaners beginning at the inner end of the gallery and with the jets and ventilating current sweeping the dust before them. In mine entries or headings it is much more difficult to arrange for placing the dust, because frequently the floor, ribs, and roof are damp. The roof and ribs are uneven except in the concreted sections, which form a small part of the whole mine. The floor is naturally rough on account of the track and ballast.

In the Altofts gallery, in England, the coal dust was placed on light temporary shelving of one-half by 5 inch boards, which rested on brackets without fastenings. Whenever there was an explosion the shelving was torn down and broken to pieces. This shelving, however, was inserted along only 500 to 600 feet of gallery. In much longer passageways, like those of the experimental mine,

replacement after each experiment would be too slow and too expensive.

For the experimental mine it was decided to use fixed and somewhat permanent wooden shelving (Pl. II, A) which could be kept reasonably dry and be easily cleaned. The shelving is made heavy, of 3 by 4 inch hard-pine material, in order to lessen breakage. It was thought that if the shelves were set close to the ribs and were continuous and smooth, like the rifling of a gun, they would not be seriously damaged. This generally has proved to be the case, except that shelving in front of an opening is liable to be broken down. The shelving is placed with the 3-inch side horizontal. In the lined sections bolts were set in the concrete wall during construction, so that each 16-foot length of shelving is supported by three $\frac{3}{4}$ -inch bolts, which pass through the timber; the bolts are drawn up, pressing the shelving tightly against the concrete. (See fig. 5.) In the unlined portions of the entry 8 by 8 inch posts are used to support the shelves (Pl. II, A); they are recessed into the ribs so as to present little or no projection for the pressure wave to strike. However, in a few tests some of the posts, in spite of being wedged, have been drawn out of the recesses, presumably by the depression wave following the explosion waves. It has therefore been necessary to anchor the posts to the ribs by bolts set into drill holes in the coal rib and cemented.

At the time of the first two official tests the shelving arrangements had not been completed, so that it was necessary to set up temporary props and place the shelving without fastenings. The result was that in the tests all the shelving throughout the mine was thrown down; some of it was broken, and many pieces were blown out of the mine to the hillside opposite.

INSTRUMENTS USED IN THE FIRST SERIES OF TESTS.

In the study of coal-dust explosions it is desirable to have flame and pressure records at as many points along the course of the explosion as practicable. It is important that each record should show the rise and fall of each pressure or depression wave that passes the recording instrument; it is equally necessary that each record should have the same time intervals marked upon it, so that the records made at successive points may be compared; and it is of the utmost importance that the position of the forward point of the flame and its duration with reference to the curves of the pressure waves shall be obtained at the various stations. The instruments used at Liévin photograph the flame directly on the manometer films on which the pressure records are photographically recorded. The instruments used at Altofts and at the experimental mine in this series of tests

record the position of the flame by a separate apparatus; therefore, to obtain a comparison it is necessary that the flame records have the same time intervals as are registered on the pressure manometer records. The French method has the obvious advantage of the records being made by the same instrument.

It was intended before the first series of tests was started to have all the various kinds of instruments used in the foreign testing galleries, and, if possible, to design and make new instruments better adapted for the particular conditions at the experimental mine, but circumstances prevented getting any but the British coal-dust instruments, like those used at Altofts. The investigations at Altofts and at Liévin were conducted in galleries above ground, so that there was little difficulty in the attachment or placing of delicate recording instruments, which could be set up on the exterior of the gallery and connected through openings in the walls. In using instruments in the experimental mine it is necessary to protect them from the action of the pressure and the flame from an explosion by placing them in special chambers (Pl. V, A, and fig. 7) as nearly gas-tight as possible, lined with reinforced concrete, and located in the side of the entry.

RECORDING MANOMETERS.

For recording the pressures three B. C. D.^a manometers similar to those designed for the experiments at Altofts were purchased. Two of these were used for high pressures and one for pressures below atmospheric pressure. These manometers are somewhat similar in principle to a steam-engine indicator and are described in detail in the chapter entitled "Description of Instruments for Measurement of Velocity and Pressure." The connections from the instruments to the main entry are made through a heavy ribbed steel-plate casting $1\frac{1}{2}$ inches thick, which is designed for a pressure of 500 to 600 pounds per square inch. The casting is set in the concrete wall of the chamber flush with the side of the entry. The instruments are placed on a wide shelf, with sufficient space behind them to allow manipulation. The cables carrying the wires for the electric circuits are in a 4-inch pipe behind the concrete lining of the entry and enter the instrument chamber under the shelving. Access to the chamber from the entry is had through a side passage. The entrance door is a heavy steel casting, designed to withstand a pressure of 500 to 600 pounds per square inch. The construction of a station is shown in figure 7.

Four instrument stations of the kind described were constructed, two in the diagonal heading and two in the main entry. These stations are at 100-foot intervals. For the next series of tests addi-

^a See p. 57, footnote.



A. INTERIOR OF INSTRUMENT STATION, SHOWING B. C. D. MANOMETER.



B. DÉBRIS ON HILLSIDE OPPOSITE MINE ENTRANCE.

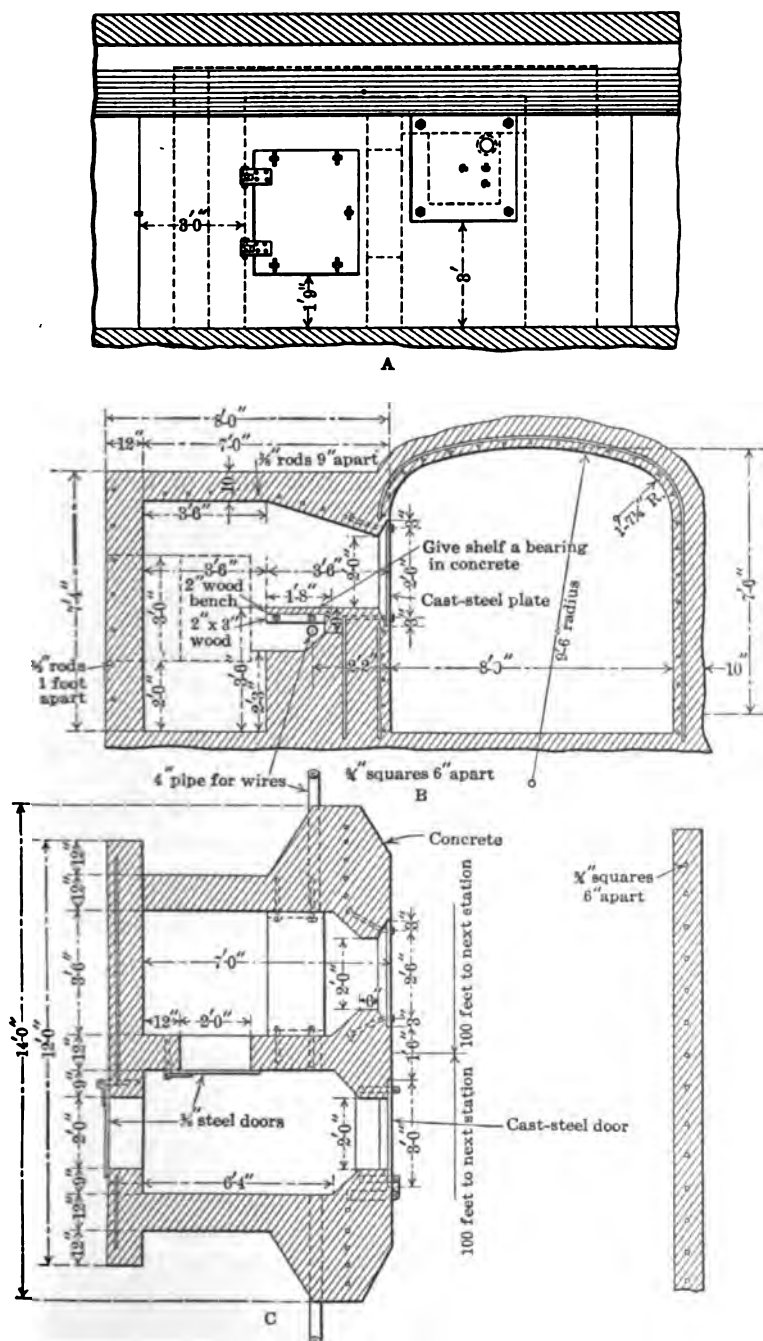


FIGURE 7.—Construction of instrument chamber: A, front elevation; B, vertical section; C, plan.

tional instrument stations will be constructed, probably at intervals of 200 feet.

MAXIMUM-PRESSURE GAGES.

In the later tests of the series the recording manometers were supplemented by maximum-pressure gages.^a They are similar to those used by Taffanel at Liévin, being of the type employed in this country and abroad in gunnery tests. The maximum-pressure gages are placed in boxes with iron-plate fronts, into which the gages are screwed, and are set in the side of the entry at 100-foot intervals. The explosion pressure acts through a steel plunger upon a small copper cylinder or column, the relative compression or shortening of which measures the maximum pressure at that point. As the copper cylinders or columns furnished with the pressure gages had not been made with sufficient care, the records obtained in the first series of tests were unreliable. It is expected that gages of better construction will be used in the future.

VELOCITY-RECORDING INSTRUMENTS.

The velocity of the flame and that of the pressure wave of the explosions were measured by the automatic recording on the chronograph of the respective times at which the flame or the pressure wave passed each station. The stations were at 100-foot intervals. The recording was accomplished by placing in the stations pressure circuit breakers and flame circuit breakers. The former operate by the action of pressure on a plunger, breaking a circuit. Two different types of flame circuit breakers were used—the detonator type and the tin-foil type. They are fully described in a subsequent section, entitled "Description of Instruments for Measurement of Velocity and Pressure."

The circuit breakers above described are placed in the rib in air-tight concrete boxes with hinged steel doors (Pl. VI, A). The circuit breakers are attached to the doors, so that when the doors open they may be readily inspected and connections made to wires leading to the outside observation station.

ELECTRIC CABLES.^b

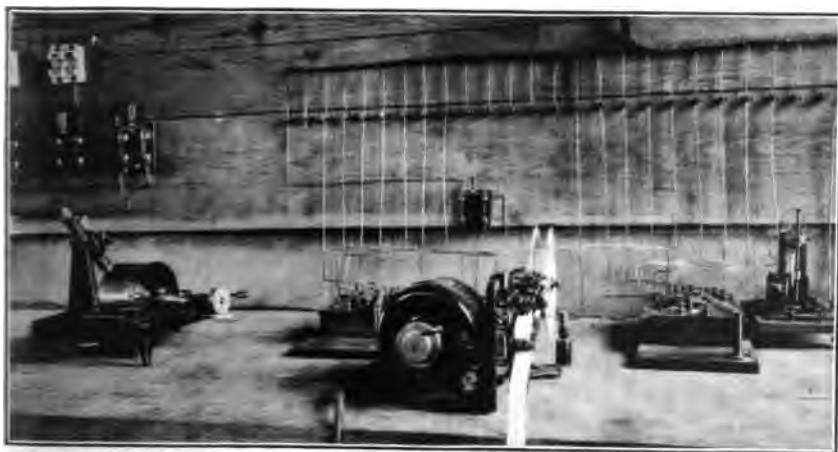
The cable carrying the wires to the various electrical instruments runs through an iron pipe set in a deep groove in the coal. The groove is faced with concrete to provide protection for the pipe and support for the coal rib. The cable is cut at each instrument chamber, and the wires are connected to binding posts on the switchboard at the back of the chamber.

^a Described on pp. 59, 60.

^b See also discussion under section entitled "Description of Instruments for Measurement of Velocity and Pressure."



A. CIRCUIT BREAKERS IN SMALL STATIONS.



B. OBSERVATORY WIRING CONNECTIONS.



C. RUPTURE OF LINING OF MAIN ENTRY.

FIRING CIRCUITS.

Among the other wires of the cables are a pair of firing wires that can be connected to form a circuit in the observatory and thus discharge the igniting shot. In the first two official experiments this wiring was temporary, but subsequently when the flame and pressure circuit wires were installed the shot-firing wires formed a part of one of the cables passing through the pipe in the rib.

The shot-firing wires are separated from the other wires at the mouth of the mine and enter a locked switch box.^a This box contains switches, which are always open until everyone has left the mine.

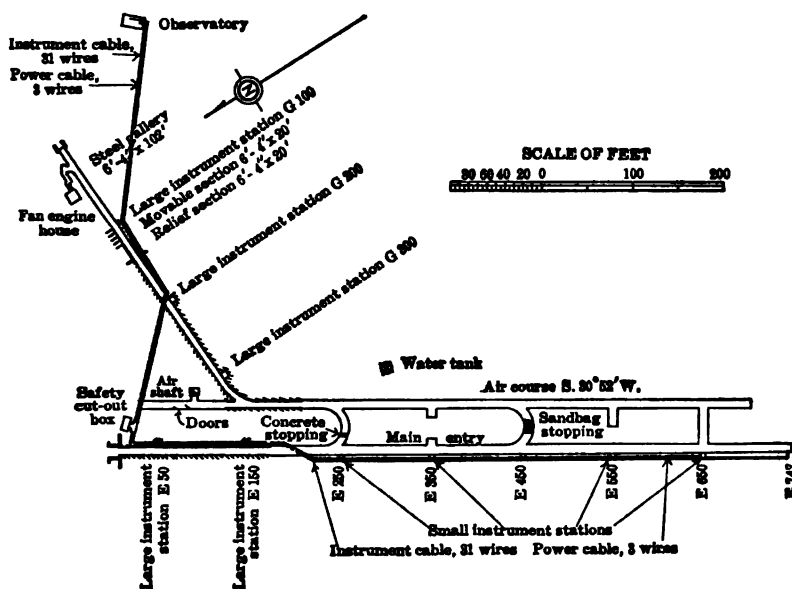


FIGURE 8.—Electric wiring of mine.

At the observatory there is another switch, also in a locked box. This switch is closed by the engineer in charge of the experiment after he has thrown in the switch at the mouth of the mine; that is, he carries the keys to both boxes, so that he may make sure that everyone has left the mine before he throws in the switches. When the switches are both in, the firing circuit is completed by pressing a button that causes the shot in the mine to be discharged. The details of the wiring are given in a subsequent section of this bulletin and a diagram of the wiring is shown in figure 8.

^a After the first series of explosions had been completed, with a view to greater safety the installation was changed. Now all the wires enter the locked box and are arranged so that all the circuits are cut by switches when men are in the mine.

CHEMICAL ANALYSES.

In order to study the chemical processes accompanying an explosion, analyses of the coal dust used and of the residual solids left after the explosion—that is, coked dust and condensed hydrocarbons—are required so that the changes in the composition of the dust and the loss in gaseous constituents may be determined. Representative samples of the coal dust may easily be taken previous to an experiment, but it is not so easy to obtain average samples of the solid residues of the explosion, and it is practically impossible to recover the total quantity of the residues.

AFTERDAMP SAMPLING.

Samples of the afterdamp may be obtained some minutes after the explosion by men wearing oxygen helmets such as are used in rescue work. It is important, however, to obtain gas samples at intervals during the explosion as well as immediately after the explosion.

This can be accomplished only with the use of automatic apparatus, like that used in the investigations at Altofts and at Liévin.

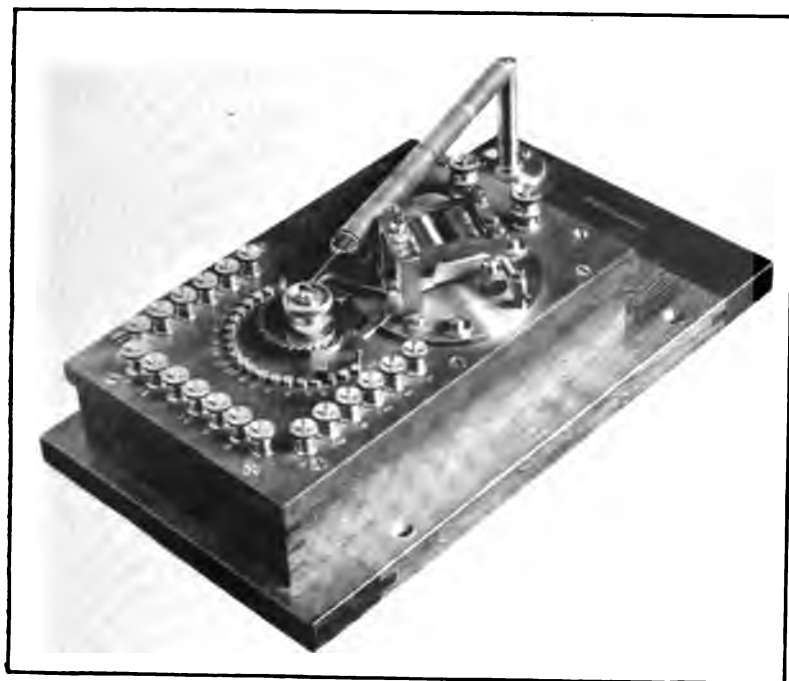
The B. C. D. automatic sampler^a (Pl. VII, A) used in the first series of tests at the experimental mine was opened by a circuit maker operated by pressure. On account of the difficulty in predetermining accurately the relative velocities of the pressure wave and the flame, the attempts during the first series to obtain gas samples during the passage of the flame were not successful. Also, in some tests the projecting pipe connections were carried away by the force of the explosion. It is planned in future experiments to have gas samplers operated by means of the flame circuit makers, to have no projecting parts, and to use devices for quickly closing the sampling bottles after the sample has been admitted.

A new type of automatic gas sampler is being developed at the Pittsburgh station for taking a set of three successive samples during or following the passage of the flame. The sampler is set in operation by a flame circuit breaker. The first sample is taken instantaneously; the others are taken at predetermined intervals by means of clockwork mechanism.

^a See full description under "Description of Instruments for Measurement of Velocity and Pressure."



A. B. C. D. AUTOMATIC GAS SAMPLER.



B. B. C. D. AUTOMATIC COMMUTATOR.

GENERAL DESCRIPTION OF EXPERIMENTS.

PRELIMINARY EXPLOSION TRIALS.

The original plan was to make explosion experiments at intervals while the mine was advancing, the explosions being originated in the exterior gallery. In other words, the ignition of coal dust was to be accomplished in the gallery in a manner similar to that employed at the Pittsburgh gallery, namely, by simulating a blown-out shot of black powder or dynamite at the closed end. However, the farther end of the Pittsburgh gallery was open, whereas at the experimental mine the farther end of the gallery, although unobstructed, pointed directly into the mine. The cannon for simulating the blow-out shot was placed against a plank stopping that closed the outer end, and a hole was cut through the stopping opposite the bore of the cannon.

Coal dust was placed along the floor of the gallery. When the large relief opening, or the movable section for connection to the mine was rolled to one side, there was no difficulty in obtaining ignitions similar to those at the Pittsburgh gallery. When, however, the large openings were closed, the effect was striking. The plank stopping was burst outward by the effect of the shot and the dust was inflamed only in the first half of the gallery. On strengthening the stopping, the flame was projected feebly through the gallery, but died away on entering the mine. Force was manifested in the vicinity of the cannon, breaking some of the heavy planks of the stopping and throwing down a sandbag stopping in the branch to the fan, but the force in the direction of the mine diminished rapidly. Seemingly the concussion of the blown-out shot did not bring the coal dust into suspension at a distance from the cannon, and the pressure developed by the ignition of the dust near the cannon, being relieved by the breaking out of the stoppings, was not sufficient to cause pressure waves large enough to continue to raise in advance coal dust in quantities sufficient to sustain propagation.

EXPLOSION TEST 1.

Several attempts of the foregoing sort were made and resulted in considerable damage to the temporary fan casing. As the time was getting short for preparing a public demonstration in connection with the first national mine-safety demonstration on October 30-31, it was

decided on October 24 to see whether a blown-out shot at the face of the main entry would start an explosion. The result was surprisingly successful. Flames shot from every opening, and there was a loud report. A car standing 50 feet from the mouth of the entry was blown off the dump, evidently being lifted over a gasoline locomotive, as indicated from certain marks on the locomotive. The locomotive was moved 15 feet. The details of the experiment are reported on pages 66 to 69. The intensity of the explosion was a surprise to the observers. Considerable damage was done to the mine, including cracking of the concrete in the main entry and in the air course at the junction of the diagonal heading. There had been a 6-foot sandbag stopping in the air course at this point, the sandbags being tightly wedged in. As the explosion came outward on the air course, the rush of the gases was momentarily arrested by the stopping, which probably acted like a water hammer to produce great increase in pressure, causing the concrete roof to be lifted and to lift up, with it the upper part of the corner pier. A piece of board and some brattice cloth were blown into the break as the roof lifted, and when it dropped it nipped the cloth and stick, as shown in Plate VIII, A. Meantime the sandbag stopping had been thrown outward, some of the bags to the outside of the mine. After the rush of the hot gases to the outside, there was a return wave to fill the partial vacuum; this carried a few of the sandbags in by their former position.

The pressure manometers at stations 150 and 50 feet, respectively, from the mouth of the mine indicated a velocity of the pressure wave between these stations of 1,954 feet per second. In this test the manometers were equipped with 30-pound springs. The pressure at station E 150^a exceeded the capacity of the spring, and the record was interrupted by the stylus being thrown off the smoked paper. The actual pressure at station E 150 was probably greater than 50 pounds. The highest pressure recorded at station E 50 was 40 pounds.

The chief lessons to be learned from this first formal test were:

- (1) That an explosion could be obtained by a single blown-out shot at the face;
- (2) That the reinforced-concrete lining was not sufficiently strong in itself to prevent rupture when an explosion approached the detonating stage;
- (3) That the instrument stations were satisfactory in their construction;
- (4) That the general arrangement of the mine was very satisfactory;

^a Text and figure references to stations E 50, E 150, etc., and to stations G 200, G 300, etc., refer to stations placed in the entry or in the gallery at points distant from the mouth of either the number of feet designated by the figures attached to the initial.



A. RESULTS OF TEST 1 AT JUNCTION OF SLANT AND AIR COURSE.



B. BENT $\frac{1}{4}$ -INCH STEEL BAR, SHOWING LIFT OF ROOF.



C. REINFORCING ROD PULLED OUT OF CONCRETE IN ROOF OF MAIN GALLERY.

(5) That temporary shelving for coal-dust load was unsatisfactory because its replacement causes serious delays in conducting consecutive experiments.

EXPLOSION TEST 2.

Although it was a rainy day, probably 1,500 people visited the mine to witness explosion test 2 on October 30, 1911. Nearly 1,200 people went through the mine previous to the test. Unfortunately delay was experienced in discharging the igniting shot prepared at the face of the main entry. This delay was due to some short circuits in the shot-firing lines, probably caused by visitors accidentally trampling on the wires. Eventually a temporary line was carried into the mine and the blown-out shot was fired. The resulting explosion was most spectacular, happening as it did on a dark, rainy night. The flame was of great extent and most vivid, and the concussion was felt in villages 4 miles distant; it is claimed to have been heard at Monongahela City, 12 miles distant.^a

Details of this test are given on pages 69 to 78.

The motors driving the manometer drums failed to start when the switch was thrown in at the observatory, so that no record of the pressure wave was obtained; only the maximum pressures were indicated, which were 26.5 pounds per square inch at station E 150 and 21.3 pounds at station E 50. There is great doubt whether these values represent the maximum pressures. The demonstrations of violence were apparently greater than in the previous test, in which the manometer readings showed higher pressures.

The lessons to be drawn from this experiment are:

(1) Coal-dust explosions may occur when the atmosphere is nearly saturated (the relative humidity within the mine was over 90 per cent) and the floor, ribs, and walls are damp, but not wet, provided the coal dust is dry and sufficiently abundant.

(2) Temporary shelving, being easily wrecked, is too expensive.

(3) There were too few recording instruments to obtain the information necessary to study satisfactorily the phenomena of a dust explosion.

(4) The magnitude of the lateral pressure of the main explosive wave of a coal-dust explosion was demonstrated by the 13-inch lift of the concrete arching in the main entry near station E 50, and by the lift of the arching in the diagonal heading, which for its entire length of over 200 feet was lifted bodily, the reinforcing rods being pulled through the concrete. This lift was about 12 inches, judging from one piece of reinforcement that was doubled back on itself (Pl. VIII, B). Not only did the strength of the lining have to

^a An account of this explosion is given in Bull. 44, Bureau of Mines, entitled "First National Mine-Safety Demonstration, Pittsburgh, Pa., Oct. 30 and 31, 1911," by H. M. Wilson and A. H. Fay, with a chapter on "The Explosion at the Experimental Mine," by G. S. Rice, 1912, pp. 29-41.

be overcome, but also the weight and resistance to shearing of 6 to 12 feet of natural strata, shale and clay, overlying the arching.

The breaking of the lining of the diagonal heading was a temporary misfortune, as it prevented resumption of experiments in originating explosions at the mouth of that heading, both because of the weakness of the cracked lining and because the breaks allowed ground water to run in freely, keeping the gallery wet. So much damage was done in the mine that several weeks was needed to make repairs. While the repairs were being made, the permanent shelving was put in place, the small stations installed for the circuit breakers, and the permanent wiring put through a line of 4-inch pipe to the head of the main entry. For safety, the pipe was set in a groove in the rib. With the small force of laborers permitted by limited funds, these repairs and improvements were not completed until the latter part of January, 1912, when the mine was put in readiness for further tests.

EXPLOSION TEST 3.

In order to test the new permanent shelving it was decided to first try explosions limited in length and, if these were successful, to increase successively the length and intensity of the explosions. The coal-dust zone in test 3 was therefore made only 84 feet long—measured from the face in which was the blown-out shot to the outer end of the zone.

It was also thought advisable in this series of experiments to make some preliminary tests of stone dust placed on shelves. Such experiments were made at the Altofts gallery, England, also independently and more fully by Taffanel at the Liévin gallery, where a special arrangement of them is termed *l'arrêt-barrage* ("arresting barrier"). In consequence of the success of Taffanel's tests, the plan of using the barriers has been generally adopted in French mines. The "arresting barrier" is composed of 10 or more shelves about one-half meter (20 inches) wide. Finely ground rock or shale dust is heaped upon them. The shelves are placed sufficiently high to give clearance for the cars, but a small space is left between the heaped-up dust and the roof, each shelf and the heaped-up dust occupying about 10 per cent of the cross section of the entry. The arrangement is such that when an explosion strikes the barrier the shelves are knocked down or demolished and the dust thrown into the air, forming what has sometimes been termed "a dust curtain," which in the Liévin gallery has been found to check or stop the explosion.

For test 3, made January 31, 1912, a set of "arresting-barrier" shelves was placed in the air course immediately outby the inmost cut-through, which was left open for ventilation. As the coal-dust loading was limited, the explosion or inflammation was small and with-

out violence. The effects on guncotton tufts suspended at intervals along the entry indicated that the flame traveled over 159 feet from the point of origin, or nearly double the length of the coal-dust zone. The branch flame that extended through the open cut-through was 63 feet long, and did not extend through the barrier. The latter result, however, was not conclusive, as the stone dust was very little displaced. There were no indications of violence. For details of this test see pages 78 to 80.

EXPLOSION TEST 4.

Test 4 followed test 3 on the same day (Jan. 31, 1912), the object being to determine whether there was sufficient unaffected dust left from the previous explosion to reignite and propagate the ignition. It was noted that the bulk of the dust had not been displaced from the shelving. No new dust was placed on shelves, but some fresh dust was placed in front of the igniting dust, and the experiment was made under practically the same conditions as present in test 3. The results were practically identical, the flame probably going about the same distance.

EXPLOSION TEST 5.

The object of test 5, made February 2, 1912, was to determine what pressures should be placed on the circuit breaker so that the record of the pressure wave from the dust explosion would not be influenced or affected by the wave from a blown-out shot of 2 pounds of black powder.

As the pressure resulting from a coal-dust explosion is at first low, it is desirable that the circuit breakers be adjusted to operate at low pressures. If, on the other hand, they are adjusted to release at too low a pressure, they will be operated by the pressure wave set up by the blown-out shot. In test 5 no coal-dust load was used, and the entry was thoroughly wetted. In order to be assured that the result would not be influenced by the presence of methane, the fan was shut down for 14 hours preceding the test, and samples of air were taken at the face before the fan was started. The samples contained only 0.06 and 0.04 per cent of methane. The sample of mine air taken at the face the following day, under normal conditions of ventilation, contained only 0.02 per cent of methane. Two pressure circuit breakers were installed at station E 650, one being set to release with a 1-pound and the other with a 2-pound pressure. Only the one set for a 1-pound pressure operated, and it was therefore concluded that circuit breakers set for a 2-pound pressure would not be operated by the pressure from a blown-out shot of the size mentioned from a cased hole in the coal.

EXPLOSION TEST 6.

In test 6, made February 3, 1912, the coal-dust load was 190 feet in length, or over double that used in tests 3 and 4. A stone-dust barrier like that described under test 3 was erected in the air course immediately outby the open crosscut. The main entry was left open. The results were that the flame, after having passed through the cut-through into the air course, died away after having passed the first two or three shelves, but the ground rock dust on these shelves was little displaced; and it is questionable how far the flame was checked by reason of the presence of the barrier. In the main entry the flame traveled about 70 feet beyond the coal-dust zone, but there was no indication of local violence.

In this test a record of the flame velocity between stations E 650 and E 550 was obtained, and showed that the distance of 100 feet was traversed in 0.7903 second, or at the rate of 127 feet per second. (See fig. 10.) For details of this test, see pages 83 to 87.

EXPLOSION TEST 7.

Test 7 was made February 5, 1912, with an increased length of coal-dust load, the length being 290 feet, or 100 feet more than in the previous test. Determination was desired of the effect of the rock-dust barrier when the length of the coal-dust zone was increased. The results showed that while the flame died away in passing through the barrier, the rock dust was little disturbed, except on the first shelf or two. In the main entry the flame traveled 465 feet, or 175 feet beyond the end of the coal-dust zone. The humidity of the air at the mine in this test, as well as in the previous one, was rather high, ranging from 93 at station E 450 to 100 per cent at station E 650 opposite the inner cut-through. The velocity of the flame between stations E 650 and E 550 was 160 feet per second. (See fig. 10.) For details of this test, see pages 87 to 91.

EXPLOSION TEST 8.

Test 8, made February 7, 1912, was virtually a repetition of the previous experiment. The coal-dust load and other conditions were the same, except that the relative humidity of the air was considerably less, being 85 per cent at station E 450 and 83 per cent at station E 650 opposite the inner cut-through. It should also be noted that the fan was shut down two minutes before the test was made, although this may not have affected the results. Perhaps the most influential cause of difference was that instead of making the blown-out shot in the coal face, as had been done in the previous tests, the ignition was caused by a blown-out shot from the cannon, which was placed at the face. The cannon probably produced a greater local concussion of

the atmosphere, and thus the explosion may have been started with greater strength. In the main entry the flame was 552 feet in length, and therefore passed 262 feet beyond the end of the zone of loading, as compared with 175 feet in the previous test. There was also much more physical manifestation of violence at the mouth of the mine.

The pressure wave between stations E 650 and E 550 had a velocity of 1,136 feet per second, and between stations E 550 and E 450 of 1,116 feet per second. This wave was probably that produced by the shot of the cannon and did not represent the explosion wave which followed it. The flame circuit breakers at the 100-foot stations between E 650 and E 250 showed velocities of 220, 254, 398, and 51 feet per second, thus indicating that the inflammation was relatively slow, and was dying away when it reached station E 250. (See fig. 10.) The last evidence of flame as shown by tufts of guncotton was 58 feet outby station E 250. Manometric pressure readings were not obtained, except for pressures below atmospheric, following the explosion. The curve obtained is shown in figure 21. The maximum depression was about 1 pound per square inch, and the total period of depression was one-fourth of a second only. The details of this test are given on pages 91 to 95.

EXPLOSION TESTS 9, 10, AND 11.

In test 8, pressure circuit breakers set to release at a pressure of 2 pounds were seemingly operated by the pressure wave from the blown-out shot from the cannon. Further tests were required, therefore, to determine the minimum pressures at which the circuit breakers might be set without risk of being operated by the wave from the cannon. In test 9, made February 8, 1912, and in tests 10 and 11, made February 9, 1912, no coal-dust load was used, and the entry was wet down with hose. The cannon was loaded with $2\frac{1}{2}$ pounds of black blasting powder, with 4 inches of clay stemming. In test 9 there was one circuit breaker at each station, and all were set to release at a 2-pound pressure; all were operated on the discharge of the shot. In test 10 two circuit breakers were installed at station E 650, one set to release at a 3-pound pressure, and one at a 4-pound pressure; both were operated by the shot. In test 11 the two circuit breakers installed at station E 650 were set, one to release at a 5-pound pressure and the other at a 6-pound pressure; neither was affected by the pressure from the shot.

As a result of these tests it was decided that in future tests when the coal dust was to be ignited by a shot from the cannon the pressure circuit breakers should be set to release at a pressure of 5 pounds. The tests demonstrated what has been suspected—that a blown-out shot from a cannon causes a higher air pressure than a shot of the same weight of charge placed in the coal face, even when a pipe liner

is used, because some of the energy of the exploding powder is expended in breaking the liner and the surrounding coal.

In tests 5, 9, and 10 tufts of guncotton were suspended at intervals along the entry. All of them within a distance of 40 feet from the cannon were ignited, probably not by flame but by burning particles of powder projected by the shot. This conclusion was reached as the result of tests in the Pittsburgh station gallery.

EXPLOSION TEST 12.

In test 12, made February 10, 1912, a lighter coal-dust load was used than previously, the quantity being 1 pound per linear foot for the first 290 feet, and less than one-half pound (0.457 pound) per linear foot for the outer 420 feet.

The flame did not travel to the outside of the mine, but stopped at about 465 feet from the origin, or at 175 feet beyond the end of loading of 1 pound per linear foot. Judging from the one experiment of this character, it would appear that under the existent conditions 0.457 pound of dust per linear foot was not sufficient to propagate an explosion. The quantity mentioned is equivalent to about 0.122 ounce per cubic foot (122 grams per cubic meter) of air space. The humidity was high, but not extremely so, being 91 per cent at station E 650. In this test the flame did not pass beyond the barrier.

As the explosion in the main entry died away before reaching the manometer stations, the pressure produced had dropped so low that no pressure record was obtained; but following the explosion the suction manometer at station E 50 showed a depression wave lasting 0.6 second, the maximum depression being only 0.5 pound per square inch (see fig. 23). The velocity of the flame between the circuit breakers at the 100-foot stations between stations E 650 and E 350 indicated velocities of 153, 161, and 225 feet per second (fig. 10), showing that the explosion was relatively slow and feeble. The details of this test are given on pages 97 to 99.

EXPLOSION TEST 13.

Test 13 was made on the same afternoon, February 10, 1912, as test 12. The same dust was used except that 25 pounds of fresh dust was spread in front of the blown-out shot, which was prepared in the coal face, instead of using the cannon as in the previous test. The result was practically the same, the flame, as indicated by the tufts of guncotton, not extending further than in the previous test. Detailed description is given on pages 100 and 101.

EXPLOSION TEST 14.

In test 14, made February 19, 1912, the coal-dust load in the outer part of the mine was increased over that used in the previous tests;

throughout the mine it averaged about 0.93 pound per linear foot, equal to 0.248 ounce per cubic foot (248 grams per cubic meter). No preliminary watering was done.

The result was a sharp explosion, the flame issuing from the mouth of the entry with manifestations of violence. In the air course the explosion entered from the inner cut-through with considerable force, dislodged most of the shelving, and blew off the rock dust. This barrier, as in the previous tests, was located immediately outby the inner cut-through. The striking feature was that the flame did not pass beyond the barrier.

The motors driving the drums for the manometer failed to run, so that maximum pressures only were recorded at stations E 150 and E 50, the records being 14.7 pounds and 5.6 pounds per square inch, respectively. The relatively small load of coal dust was seemingly the cause of the pressures not being higher. Both the pressure and the flame circuit breakers failed to operate at some of the stations and thus prevented velocity records from being obtained. For details of this test see pages 101 to 105.

EXPLOSION TEST 15.

As test 15, made February 24, 1912, was the last test of the series, a public demonstration had been planned; consequently a number of State inspectors and other mining men were present. The coal-dust load per linear foot was double that previously used, averaging 2.13 pounds per linear foot, equal to 0.568 ounce per cubic foot (568 grams per cubic meter). It was also more extensively distributed than in any other experiment except test 2; it was spread throughout the main entry and two-thirds of the way in the air course. The rock-dust barrier shelving was changed from its position in the previous experiments to a point in the air course about halfway out in order to give it a more thorough test.

The result of the experiment was that the flame following the cloud of dust shot out of the main entrance accompanied by a loud, sharp report, and with manifestations of great violence. No flame appeared from the air-course entrance; the rock-dust shelves were demolished; the flame passed through the barrier, and traveled about 200 feet beyond, as shown by the ignition of tufts of guncotton suspended at intervals along the entry. It probably ignited the coal dust in a 50-foot zone that laid outby the barrier. In view of the great pressure existing within the air course, as demonstrated by the demolition of the barrier, it is quite probable that the flame might have reached the outside but for the rock dust dislodged from the barrier. However, the result as to the efficacy of the barrier must be considered inconclusive. A graphic representation of the probable extent of the flame is shown in figure 28.

Interesting, though incomplete, information as to the velocity of the pressure wave and of the flame was obtained from the instrument records. The pressure circuit breakers between stations E 650 and E 350 indicated velocities of 1,139, 48, and 980 feet per second. The first value, 1,139, probably represented the speed of the wave from the blown-out shot, and not of the explosion itself; the second value, 48, obtained by difference between the registration of the shock wave and the next registration by the explosion wave, must be disregarded; the record between stations E 450 and E 350 indicates that the pressure wave from the explosion had fairly started. (See figs. 9 and 11.) Farther out the pressure circuit breakers failed to work. The manometers at stations E 150 and E 50 indicated that the pressure wave had moved between the stations at the rate of 800 feet per second. The maximum pressures shown by the manometers were higher than previously obtained, the manometer at station E 150 indicating 103 pounds per square inch, and at Station E 50, 61 pounds. It will be noted in figure 27 that the pressure curves are incomplete, so that it is possible that higher pressures were attained than shown by the marks of the recording stylus. Following the explosion, the suction manometer at station E 50 recorded a depression lasting 3 minutes, with a minimum depression of 1.8 pounds per square inch. (See fig. 26.)

From the instant of ignition at the face until the pressure wave reached station E 350 the time interval was 2.367 seconds; the flame traveled the same distance (397 feet) in 2.537 seconds. The instruments indicated (if their relative lag be disregarded) that the flame at station E 350 was only 0.17 second behind the front of the pressure wave that acted on the pressure circuit breaker. (See fig. 9.) The successive readings of the flame circuit breakers between stations E 650 and E 450 were 408 feet and 1,351 feet per second; between stations E 450 and E 250 the tremendous velocities of 2,273 and 2,128 feet per second were indicated by the apparatus. Outby station E 250, on account of the failure of the automatic commutator to operate, records were not obtained. It would appear that the explosion was slow at the start but that by the time it had reached station E 450 it had approached a "detonating stage."

The most striking features of this experiment were the comparatively slow development of the explosion, the relatively slow speed of the pressure wave, and the rapid overtaking of the pressure wave by the flame of the explosion.

COMMENTS ON THE DATA OBTAINED.

Although the tests were too few, and the data obtained too meager, to admit of final conclusions being formed concerning the phenomena of coal-dust explosions, the results are of value in an educational way in demonstrating to the mining public the explosibility of coal dust when fire damp is not present, and the violence and destruction that accompany extensive coal-dust explosions, even when fire damp was not present. They were also useful in furnishing information to those conducting the investigation concerning the conditions under which coal-dust explosions may be brought about in the experimental mine, the kind of equipment required, and especially the character of instruments and apparatus necessary for properly recording and studying the phenomena of such explosions.

RECORDS OF PRESSURE VELOCITIES.

In the course of the experimental tests there were obtained records of five pressure waves in three experiments that were clearly "shock" waves started by the igniting shot. Two of these were registered when coal dust was not used, so that there is no question as to the correctness of the interpretation of their being waves started by the shot and not by the explosion of coal dust.

The velocities recorded were: Test 8, 1,136 and 1,116 feet per second; test 9, 1,111 and 1,099 feet per second; test 15, 1,139 feet per second; average, 1,120 feet per second.

The ventilating current at the time of the passage of the shock wave had in each case a velocity of about 2 feet per second in the same direction; deducting this, the net average velocity of the shock wave was therefore 1,118 feet per second, as shown by the combination of circuit breakers, automatic commutator, and chronograph.

The temperature of the air in the passageway during these tests averaged 42° F. A sound wave travels through air at this temperature at the rate of 1,096 feet per second.^a A "shock" or impulse wave from the cannon, transmitted through the air, is equivalent to, or has practically the same longitudinal velocity as, a sound wave under the conditions through a tube or gallery. The difference, only 22 feet or 0.02 per cent, indicates the substantial accuracy of the pressure-velocity instrument system; on the other hand, dependence

^a Landolt-Börnstein, *Physikalisch-chemische Tabellen*, 1905, p. —, results of Ciccone and Campanili.

on a single-stylus chronograph (time marker) and an intermediate automatic commutator which, under the arrangement of the instruments, did not act quickly enough for the higher velocities, was not satisfactory, as a failure of any one of the circuit breakers to operate prevents obtaining records beyond that point. Therefore, it has been decided that in future tests there is to be a chronograph with as many recording pens as there are circuit breakers, so that each may be

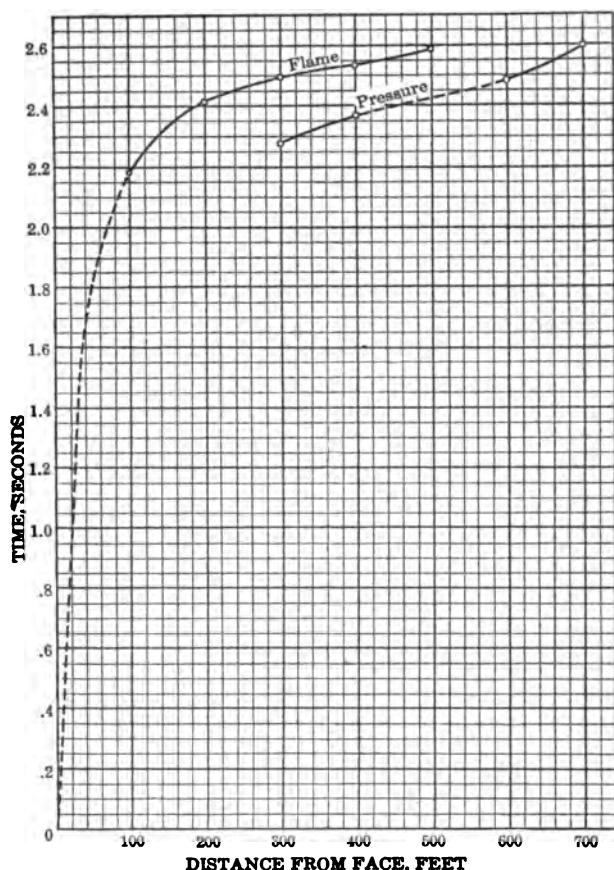


FIGURE 9.—Relative positions of flame and pressure wave, test 15.

independent in registration although marking on the same drum; the registration will then be referable to the same time intervals.

When a blown-out shot has been fired the shock wave raises the coal dust in the vicinity into the air, and the flame ignites the dust. At first it would seem there is merely quiet combustion, but if the coal dust is inflammable, pressure is gradually developed and the pressure waves start. In test 15 the period of developing pressure, including time of traversing the first 300 feet, occupied $2\frac{1}{2}$

seconds from the time of ignition. When the pressure wave reached the 300-foot point it was far behind the shock wave caused by the blown-out shot. The pressure-velocity curve obtained from the records of test 15 is shown in figure 9. Although the data are incomplete, sufficient details were obtained to make apparent the slow development of pressure and velocity to the 300-foot point, where the pressure wave started off at a higher rate but less than the velocity of a sound wave. At the 600-foot point the manometers indicated a slowing up between the stations at 600 feet (E 150) and 700 feet (E 50) from the origin. It is possible that the indicated decrease in velocity is due to the lag of the manometer at the 700-foot point. Much additional information from further experiments must be obtained before conclusions can be drawn as to the normal pressure-velocity curve resulting from uniform dust loading in a given size and shape of passageway, and other known controlling factors.

RECORDS OF FLAME VELOCITIES.

The data of indicated flame velocities in the first series of tests are more satisfactory than those of the pressure velocities. There are fewer difficulties with the flame circuit breakers than with the pressure circuit breakers, since there is only one flame record to be made; but there was difficulty with the automatic commutator and single-point chronograph, since failure of any breaker meant loss of all subsequent records.

Figure 10 shows the curve plotted from the flame-velocity records from tests 6, 7, 8, 12, and 15. These curves indicate that for the first 200 feet the velocity is relatively low, averaging less than 200 feet per second. In the case of explosions showing the least violence, the velocities did not materially increase with distance, though in test 8 the velocity reached 400 feet per second. This velocity was attained about 60 feet beyond the end of the coal-dust zone; but, as a large amount of coal dust would be carried forward by the wave itself, there would be no lack of fuel for considerable distance.

Test 15 was most complete and satisfactory, although unfortunately the velocity records were not obtained as far as the mouth of the mine. A feature of great interest in the flame-velocity curve (fig. 11) plotted from the record is the seeming attainment and passing of a maximum velocity between the points 350 and 400 feet from the origin. However, as there was only one registration beyond this apex, it would not be safe to assume that a normal flame-velocity curve drops after a maximum has been reached.

In figure 9 are shown the time intervals from the moment of ignition of the blown-out shot to the instant of passing the various stations of the flame and the pressure wave, respectively. The last two points on the pressure curve, corresponding to time ordinates 2.48 and 2.605,

were obtained from the B. C. D. manometer record; the other points were obtained from the chronograph record. The curves in figure 9

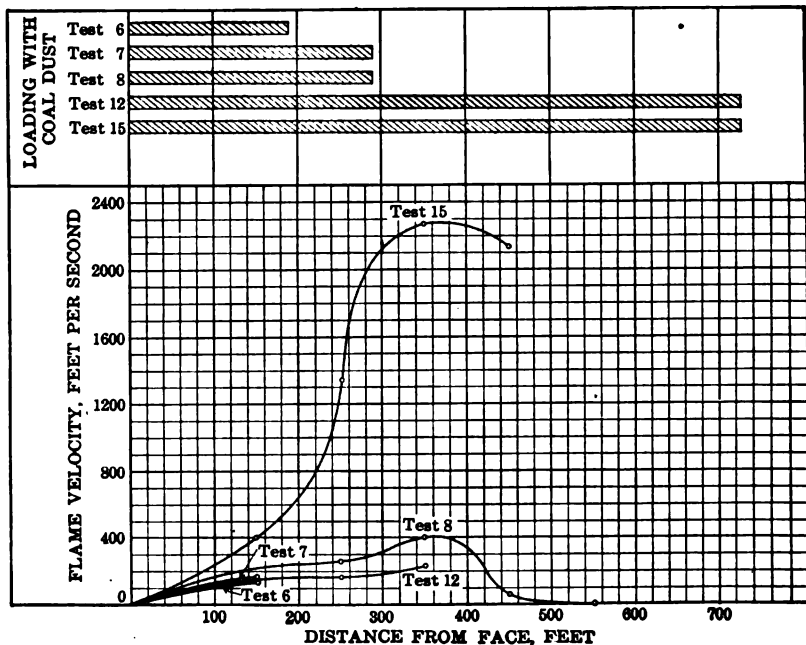


FIGURE 10.—Flame-velocity curves obtained in tests 6, 7, 8, 12, and 15.

show the relative position of the flame and that of the pressure wave, as obtained from the data from test 15. The horizontal distance



FIGURE 11.—Pressure-velocity and flame-velocity curves obtained in test 15.

between the two curves indicates the distance that the front of the pressure wave is in advance of the flame.

From the evidence of test 15 alone, the flame would appear to travel at the start more slowly than the front of the pressure wave; at 2.3 seconds after ignition the curves (fig. 9) indicate that the flame was 225 feet behind the front of the pressure wave; at 2.4 seconds after ignition it was 250 feet behind; at this point it began going faster than the pressure wave. The precision of these registrations is admitted to be somewhat uncertain. As stated, the pressure circuit breakers indicate only the front of the pressure wave; the flame circuit breakers do not act until the tin foil has melted, so that there is an appreciable lag, although this lag should be the same at each circuit breaker. From observations of issuing flame and records obtained from surface gallery explosions and from Taffanel's records of the Liévin station, it would seem that the flame is pointed, the point being in advance of the body of the flame, which fills the passage and melts the tin foil. If this is so, it is probable that the point of the flame may be much closer to the front of the pressure wave than is indicated by the curves of figure 9.

In future experiments the relation of the flame to the pressure wave will be made a special study. Manometers like those used by Taffanel will be tried. As the manometer drums carry sensitized paper the pressure curve will be photographed through the medium of a ray of light thrown on a mirror attached to a deflecting diaphragm, and simultaneously, through a glass-covered opening, the light from the flame will be photographed on the same paper, so that the exact time relation may be obtained.

RECORDS OF PRESSURES.

The B. C. D. manometers proved to be fairly satisfactory for the conditions to which they were subjected. Unfortunately, the motors supplied with the manometers were too small, with the result that in several tests no records were obtained. For the last test (test 15) stronger motors were installed. Although the instruments were designed with a view to reducing to a minimum the inertia of the moving parts, this inertia is large enough to introduce considerable errors in the records of pressures that rise with the rapidity of those resulting from explosions. In future tests it is planned to supplement the B. C. D. manometers with instruments of different design, in which the inertia of the moving parts will be much less.

As already commented, it is probable that the manometric curve (fig. 15) for station E 150 in test 1 does not show the maximum pressure. The highest point registered was 50 pounds, but it is quite possible that the peak of the curve was considerably higher. The same is true of both the manometric curves (fig. 27) in explosion 15, in which the highest point of the curve registered by the manometer at station E 150 was 103 pounds, and at station E 50, 60 pounds.

The B. C. D. pressure manometers do not register pressures below atmospheric pressure on the same instrument that records the positive pressures. In several of the experiments one of the manometers was arranged to record pressures below normal atmospheric pressure. In each case the instrument was placed in the station 50 feet from the entrance. In test 15 the maximum depression registered at this point was about 1.80 pounds, sustained for one-tenth of a second, the entire period of depression being three seconds.

MAXIMUM-PRESSURE GAGE RECORDS.

On account of imperfections in the copper cylinders of the maximum-pressure gages used in the later tests of the first series, the results obtained were not reliable. It is hoped that with improvements in making the cylinders the maximum-pressure gages will furnish as reliable a check on the records obtained with the recording manometers as that reported in the Liévin tests.

GAS SAMPLING.

The difficulties that were experienced in getting samples with the B. C. D. automatic samplers have been discussed in detail on page 38. No samples of afterdamp were obtained except those gotten by hand at a considerable interval after the explosion. In addition to the improvements planned on the B. C. D. instrument, it is proposed to employ other gas samplers arranged in groups of three at one station, to be automatically actuated by the flame so that three gas samples may be taken at predetermined intervals during and after the passage of the flame.

Two samples of afterdamp, gathered by men wearing breathing apparatus 15 minutes after explosion test 15, are of much interest. They were taken at points only 150 and 200 feet from the mouth of the main entry; consequently they were largely diluted by the air that entered immediately after the explosion. The most striking feature of the analyses (see page 113) is the high ratio of carbon monoxide to carbon dioxide (1:1.51 in the first instance and 1:1.36 in the second instance), which seems to indicate that at the time of explosion there was a great excess of carbon in the air.

VENTILATION.

In all of the experiments, except in test 8, there was employed a ventilating current with a velocity of about 100 linear feet per minute, the current traveling in the same direction as the explosion. Test 8 was made in still air. In other words, the explosion traversed the return-air passageway. In the next series of experiments it is

planned to use ventilating currents of higher velocity and to duplicate certain explosions, except in one test to have the air travel with the explosion and in the other test against it.

HUMIDITY.

It happened that in the majority of the experiments the humidity of the air was high. This was especially the case in the public demonstration on October 30, 1911 (test 2). Therefore, it would appear from these tests that the humidity of the air itself has no appreciable effect upon a strong explosion. In almost all the experiments the floor was kept moist in order to prevent the dust of the track ballast from being brought into the air and affecting the explosion. The coal dust in all cases was dry and had been placed only a few hours previous to the experiment.

DESCRIPTION OF INSTRUMENTS FOR MEASUREMENT OF VELOCITY AND PRESSURE.

In order to devise means for the prevention of coal-dust explosions, as well as for the arrest of explosions that have started, a knowledge of the nature of such explosions and of the accompanying phenomena is necessary. It is essential to ascertain the physical laws governing the propagation of the explosions and the chemical processes that take place, as well as the means by which ignition of the dust is brought about. A study of the propagation of a coal-dust explosion requires the use of instruments for recording the pressures developed at various stages of the explosion, for recording the velocity of propagation of the explosion, and for taking samples of the gases produced by the explosion.

Instruments especially adapted for this purpose have been designed in England for the experiments of the British Coal Dust Commission at Altofts, and in France for the investigation at Liévin. Most of the instruments used in the preliminary experiments described in this report are the same as those used at Altofts and were furnished by the English manufacturers.

As the experiments at Altofts were made in a steel gallery above ground, recording instruments used could be mounted outside the gallery, no special precautions being necessary for their protection. The protection of instruments for recording the results of explosions in an underground gallery is more difficult. In the experimental mine of the Bureau of Mines, protection is afforded by rooms cut in the rib at intervals along the entry and closed by steel doors (Pl. V, A). The instruments connect with the entry through openings in the doors. A description of these instrument stations appears in a preceding section of this report.

PRESSURE RECORDERS.

Two types of instruments are used for the measurement of pressure, one to obtain a continuous record of the pressure and the other to record the maximum pressure.

B. C. D. MANOMETER.^a

The B. C. D. manometer (Pl. IX, A, and fig. 12) is of the former type, and records the movement of a strong, tempered-steel, triangular spring (*s*, fig. 12) under the pressure of the explosion. The spring is clamped at its base; the force of the explosion acts vertically upward at its apex, which carries a fine steel point or scribing style. In this form of spring the maximum strain is the same at all cross sections, and the inertia is lessened, since the narrow light part of the spring moves most and the wide heavy part least.

The pressure is transmitted by an oil cushion to a cylindrical plunger (*p*, fig. 12), which can move freely in a vertical direction in an open cylinder. The oil keeps the plunger well lubricated, reduces

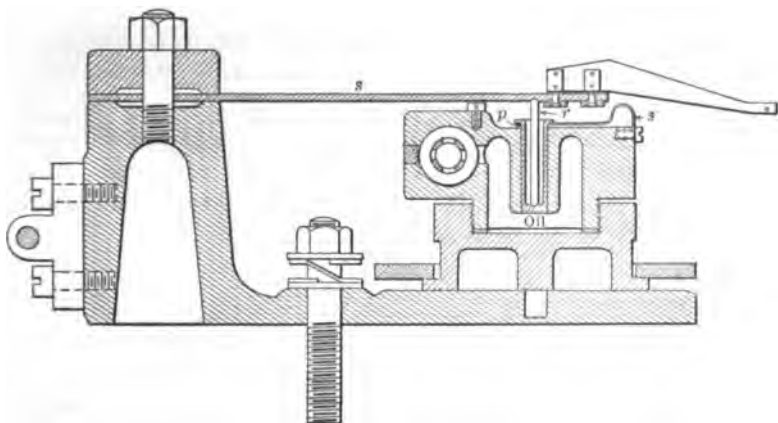


FIGURE 12.—Section of B. C. D. manometer.

leakage past it, and prevents the fine dust with which the compressed gases are charged from cutting or clogging the rubbing surfaces; the oil box also acts as a dash pot, damping any rapid vibrations. The plunger is hollow for the sake of lightness. It makes contact with the spring by means of a loose connecting rod (*r*, fig. 12), one end of which rests in a conical depression in the bottom of the plunger, the other end resting in a similar depression in the lower face of the triangular spring (*s*, fig. 12). A secondary weak spring (*s'*) acts upward on the plunger, holding the connecting rod in place when there is no pressure acting on the oil box. The scribing style on the end of the triangular spring (*s*) presses lightly against a smoked-paper band, which is lapped around a revolving drum. The drum is driven at a constant speed by an electric motor. After an experiment the smoked paper is removed and the surface fixed by immersion in a bath of transparent varnish.

^a "B. C. D.," the distinguishing mark used for these instruments by the makers, refers to the British Coal Dust Commission, for whom they were designed.

The pressure of the explosion is transmitted from the mine entry to the manometer through a piece of heavy hose, as short in each case as conditions will permit, usually 1 foot to 2½ feet long.

In order to compare one pressure record with another record of the same explosion it is necessary to know the relative position of each manometer drum at the instant the explosion started, and the speed of revolution of each drum. Two Deprez indicators attached to each manometer are used for this purpose. The Deprez indicator is a small electromagnet that holds down, against the pull of a weak spring, a light aluminum style resting against the smoked surface of the manometer drum. As soon as the electric circuit is broken the style moves in a direction perpendicular to the direction of rotation of the drum.

An indicator on one manometer is connected in series with one on the next, and the circuit goes through a fine wire stretched in front of the igniting shot. At the instant this wire is broken by the shot, each style makes a mark on the smoked paper on its drum.

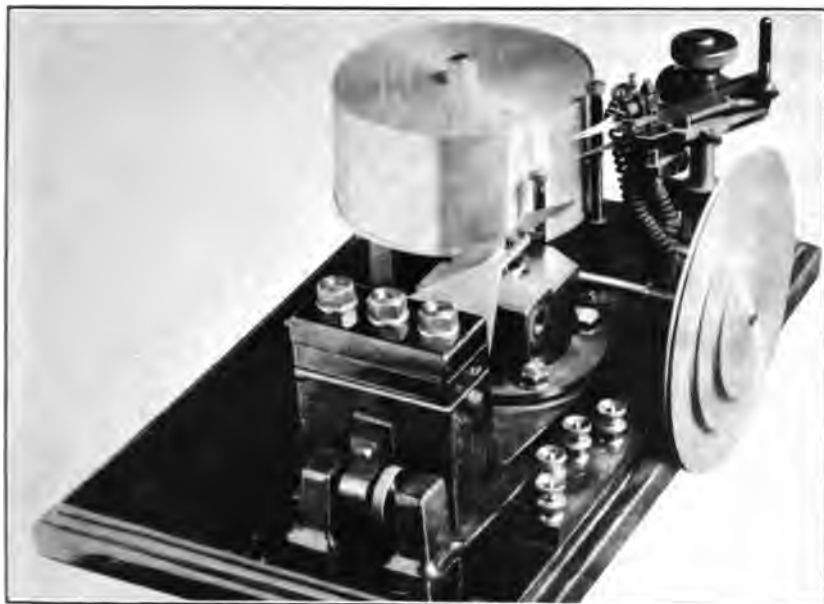
The second indicator on each manometer records the speed of revolution of the drum. These indicators are connected in series with each other and with the "tenth of a second time marker," known as the B. C. D. time marker.

B. C. D. TIME MARKER.

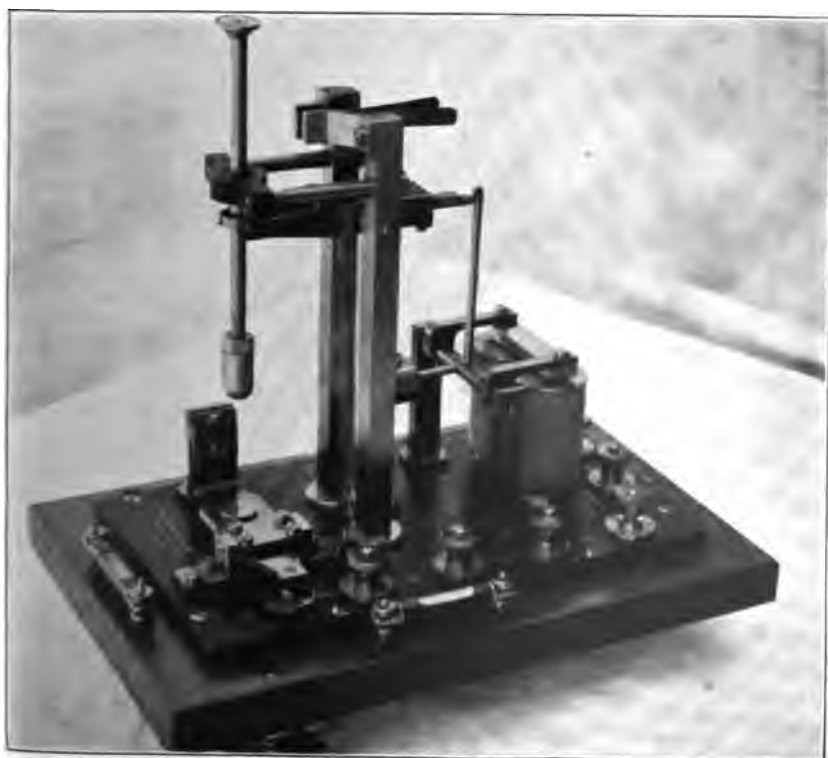
The B. C. D. time marker (Pl. IX, *B*) has a weight so arranged that when suspended at rest an electric current passes through it; when the weight falls, the circuit is broken. While the weight is falling, a circuit is again made, but is broken when the weight comes to rest. Consequently, when the time marker is connected in series with the Deprez indicators on the manometers, a break, a make, and a break are successively traced on the drums by the styles.

The space between the two breaks recorded on the drum is equivalent to the time required for the weight to fall from rest through the given distance. This distance, S , is calculated from the dynamic formula $S = \frac{1}{2}gt^2$, so as to represent a time interval, t , of just one-tenth of a second. The space between two breaks, instead of between a break and a make, is measured so as to eliminate any error from the lag of the indicator. This lag would, of course, be the same for two breaks, but might be different for a break and a make.

The two breaks in the circuit are obtained in the following manner: The weight rests freely on brass knife edges in a V support pivoted at the middle. When the weight is thus supported the circuit is complete and the current can pass through the electromagnets on the manometers. As soon as a catch is released a spring rotates the support from under the weight and the circuit is broken at the



A. B. C. D. PRESSURE MANOMETER.



B. B. C. D. TIME MARKER.

instant the weight begins to fall. The strength of the spring is such that the support is rotated well out of the way of the knife edges on the weight rod. While the weight is falling, the opposite end of the support flies up and closes the circuit again. As soon as the weight hits the platform at the bottom of its fall the circuit is again broken.

By means of suitable electrical connections, the catch is released by an electromagnet operated by the same switch that fires the igniting shot, so that the time interval recorded begins at the moment the explosion is started.

B. C. D. MEASURING APPARATUS.^a

The smoked paper records, after being removed from the manometer and fixed in a bath, are placed on a special drum (Pl. VI, *B*) and are examined under a microscope. The circumference of the drum is graduated into millimeters, so that the distance between any two points on the chart can be measured by rotating the drum and bringing the points on the chart in turn under the cross hairs of the microscope. Since the $\frac{1}{16}$ -second interval marked on the chart is always over 50 mm. long, 0.1 mm., which may be read on the chart, would correspond to one five-thousandth of a second.

By means of a micrometer screw with a divided head, the drum can be moved axially in the direction of motion of the pressure spring. By this means the pressure ordinates may be measured accurately to 0.01 mm.

The manometer springs are calibrated by applying known pressures to the manometer with a Schaeffer and Budenberg dead-weight pressure-gage tester and a standard gage.

SUCTION MANOMETER.

One of the three B. C. D. manometers used in this work is so made that it may be converted into a suction manometer; it was used as such during the tests of January and February, 1912. A much lighter spring is used for this purpose and is placed so that its zero position is higher on the drum than that of the pressure springs, thus allowing room for a downward deflection. The plunger, by means of a light framework, makes contact upon the upper face of the spring. Any suction applied to the manometer pulls the plunger downward, deflecting the spring in this direction. The spring is calibrated by means of a vacuum pump and a mercury manometer.

MAXIMUM-PRESSURE GAGES.

The maximum-pressure gages are of the "crusher" type used in ordnance work to measure the pressures in the chambers of large guns, and are similar to the gages used by Taffanel.

^a See Pl. VI, *B*.

A cross section of one of the gages is shown in figure 13.

The value of the maximum pressure is determined from the shortening of a small copper cylinder under pressure. A light hollow plunger (*p*) fits into one end of a cylinder (*b*), the other end of which is open to the mine entry. Grooves cut around the outside of the plunger, or piston, are filled with heavy grease or tallow to prevent corrosion and reduce leakage past the plunger. The copper cylinder, which is about 0.063 inch in diameter and 0.1023 inch long, is set on end in the bottom of the plunger, and a cap (*c*) with a projecting pin (*a*) in the center is screwed down against it.

The length of the copper cylinder is measured with a micrometer caliper before and after the test, and the pressure is determined from the amount of shortening of the cylinder.

Eight of these gages and a large number of copper cylinders were made at the naval gun factory, Washington, D. C. The calibration curve for the copper disks was furnished by the maker.

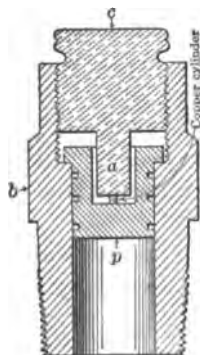


FIGURE 13.—Cross section of maximum-pressure gage.

VELOCITY RECORDERS.

The method of measuring the velocity of the pressure wave is to allow it to break a series of electrical circuits, at regular intervals along the mine. The breaking of each circuit is recorded by the release of an electromagnetic pen which makes a line on a suitable surface traveling at a known speed.

B. C. D. PRESSURE CIRCUIT BREAKER.

The breaking of the circuits at the various points is accomplished by means of the B. C. D. circuit breakers, which are adjusted so as to operate circuit only when a definite pressure has been exceeded. The breaker (Pl. VI, A) has a tube opening at one end into the mine entry and connected at the other end to a cylinder containing a plunger, which is held down by a lever and heavy spring. By varying the tension of the spring the pressure required to lift the plunger may be fixed at any value from 2 to 20 pounds per square inch. A pin projects from the side of the plunger, and when set this pin rests on two aluminum plates completing an electric circuit between them. A small spring acts on the outer end of this pin and tends to rotate the plunger; rotation is prevented by the aluminum plates, their shape being such that the downward pressure of the plunger holds the pin on both of them. As soon as the explosion raises the plunger, the contact is broken and the small spring pulls the pin to one side, thus preventing any remake of the circuit when the plunger falls again.

AUTOMATIC COMMUTATOR.

Any variation in the lag of the electromagnetic pens used to record the breaks of the circuit breakers, although short in itself, might constitute a comparatively large percentage error since the time interval to be measured is so very short.

Such an error is avoided in this apparatus by using an instrument termed an "automatic commutator" (Pl. VII, *B*, and fig. 14), which enables each successive break to be recorded by the same electro-magnetic pen. This instrument is used in connection with the recording instrument (chronograph) described on page 62. The

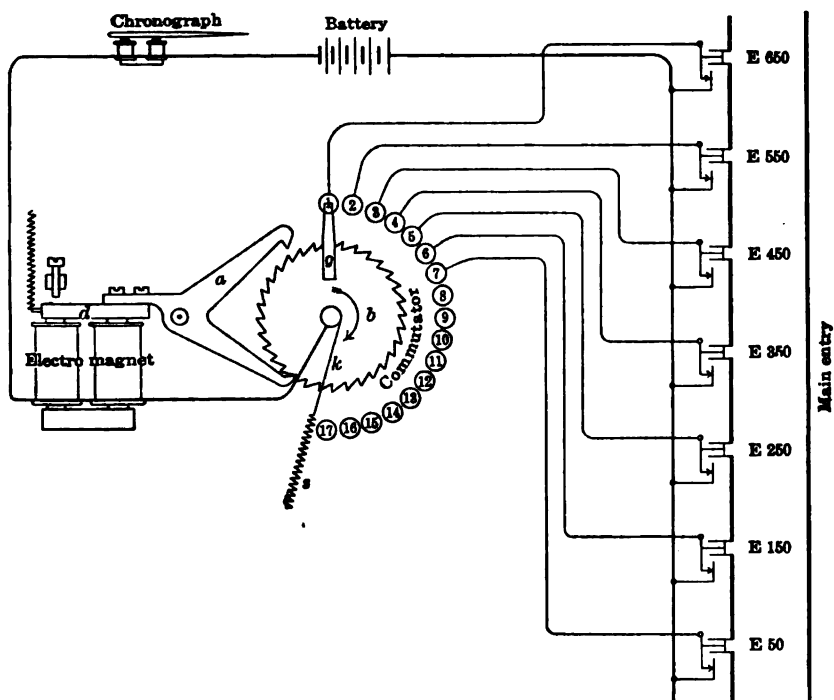


FIGURE 14.—Wiring diagram of automatic commutator.

current from the battery (fig. 14) goes through the electromagnet on the chronograph so that its armature is attracted, and then through the electromagnet of the commutator to the brush (*g*, fig. 14). The other terminal of the battery is connected to one terminal of each of the circuit breakers (at E 650, E 550, etc., fig. 14). The second terminals of the circuit breakers, in the order in which the explosion reaches them, are connected by separate wires to the studs on the commutator (1, 2, 3, etc., fig. 14).

Assume that the brush *g* rests on stud 1 (the position it occupies before the explosion); the current then flows in order through the

electromagnet on the chronograph, the commutator electromagnet, the brush *g* circuit breaker 1 (as at E 650), and back to the battery. If the explosion breaks the circuit at circuit breaker 1 (as at E 650) the chronograph electromagnet releases its armature and the pen makes a mark on the moving surface; at the same time the armature *d* of the commutator electromagnet is released and the anchor escapement *a* attached to the armature is moved. This allows the spring *s* to pull the scape-wheel *b* around by the cord *k*, which is wound on a drum, attached to the axis of the scape-wheel. The brush then moves part way to stud 2; the current immediately begins to flow through circuit breaker 2 (as at E 550), the chronograph electromagnet, and the commutator electromagnet; the armatures of both magnets are again attracted; and then the pen on the chronograph is moved back to its former position. On the commutator, the armature *d* and the escapement *a* move back to their former position and the brush *g* moves a little farther on to stud 2. When the explosion operates circuit breaker 2 (as at E 550) the brush moves to stud 3, the same cycle being repeated, and so on, for as many points as may be fixed, all the breaks in the circuits being recorded by the one pen on the chronograph so long as they occur in the proper order. It will be readily seen, however, that if any one circuit breaker fails to operate properly, no records will be obtained at the circuit breakers following, since the armature on the automatic commutator will not be released, and consequently the brush will not be free to move to the next stud.

CHRONOGRAPH.

The laboratory chronograph (Pl. VI, *B*) used in connection with the automatic commutator and circuit breakers traces a record in ink on a long riband of paper. It comprises three electromagnets to the armature of each of which is attached a brass recording pen that moves with the armature when the electric circuit is made or broken. The pens rest lightly on the strip of paper which is drawn along at a uniform rate by an electric motor. One of the electromagnets is connected in series with a storage battery and a "contact clock" which breaks the circuit every half second, thus giving the speed at which the paper travels. A second electromagnet is connected as described above, and its pen records the various breaks in circuit due to the pressure wave of the explosion. From the breaks the velocity of the wave may be determined. The third pen is used to record the travel of the flame during an explosion.

FLAME CIRCUIT BREAKERS.

The method of obtaining the velocity of the flame is the same as that for obtaining the velocity of the pressure wave, except that a different form of circuit breaker is used. The breaker must be sensitive to heat but not affected by pressure.

The first form of flame circuit breaker tried was simply a detonator, such as is used for "setting off" dynamite, placed in a wooden block, so that its open end would be exposed to flame passing through the entry and having a small wire wrapped a couple of times around it. The ends of this wire were carried back through separate holes in the wooden block and connected in the automatic commutator circuit. Each detonator was made extremely sensitive to heat by coating the inside and the open end of the brass capsule with silver carbide, which explodes at a lower temperature than the fulminate. This carbide coating was suggested by W. O. Snelling, chemist, formerly connected with the bureau.

In a slight modification of this form of circuit breaker the silver carbide was placed on a match stick instead of directly upon the brass cap. The advantage of the modification was that the match stick could be put in the detonating cap a few minutes before the explosion and not have as much time to collect moisture from the damp mine air as did the cap, which had to be in place two or three hours earlier.

The detonator form of flame circuit breakers, however, was not entirely satisfactory. It was finally replaced by the tin-foil circuit breaker, which consists simply of a small strip of tin foil that is fused by the heat. Each strip is about one-fourth of an inch wide and 1½ inches long, and is placed in a groove (about three-sixteenths of an inch deep) in a wooden block, for protection against mechanical injury. Wires fastened to either end of the strip are carried back through the block and make connection with the instruments in a manner similar to that described for the pressure circuit breakers, except that circuit breaker 1 is connected to stud 2 (fig. 14) on the automatic commutator. The first point on the commutator is connected to a fine wire stretched across the entry in front of the igniting shot. The wire is similar to the "ignition wire" connected to one indicator of the B. C. D. manometers.

The pressure circuit breakers and flame circuit breakers, in the series of tests here described, were placed at intervals of 100 feet along the main entry of the mine in the stations already described, the first one being about 100 feet from the igniting shot.

B. C. D. AUTOMATIC GAS SAMPLER.

The apparatus used to obtain a sample of gas during the explosion is of the B. C. D. automatic type (Pl. VII, A), and is designed to meet the following conditions:

(1) The exact moment at which the sample is taken with reference to the explosion must be known and must be capable of alteration at will.

(2) There must be no air in any of the connecting pipe between the sampling bottle and the mine entry.

(3) The time during which the sample is being taken must be very short if a sample from only one stage of the explosion is desired.

A steel sampling bottle is connected to a short pipe that just reaches into the mine entry. The inner end of this pipe is closed by a glass cap and the whole (bottle and pipe) is exhausted to a pressure of about 3 mm. of mercury.

A circuit maker, which is similar to the B. C. D. circuit breaker except that its operation closes an electric circuit instead of opening one, is placed at any desired location in the mine. The pressure wave causes the circuit maker to close an electric circuit through a fine fuse wire that holds back a hammer over the glass cap of the sampler. The current fuses this wire and allows the spring to pull down the hammer and break the cap; then gas rushes through the pipe and into the exhausted bottle.

The current also passes through an electromagnet in parallel with the fuse wire, so that at the same time the wire is fused the armature of the magnet is pulled down and releases a trigger which in turn allows a small weight to fall. Gas enters the bottle during the time this weight is falling. To the weight is attached a cord which, after the weight has fallen a short distance, suddenly jerks a pin that releases a heavy weight. This weight, by means of a cord and pulley, turns a valve that closes the bottle. The length of time that the bottle remains open is regulated by the length of the cord attached to the small weight.

The instant of taking the gas sample is fixed by the location of the circuit maker. Thus, if the circuit maker is located alongside of the gas sampler, the time of sampling will coincide with the passage of the front of the pressure wave. By placing the circuit maker between the gas sampler and the point of ignition, the gas sample may be taken in advance of the pressure wave.

ARRANGEMENT OF APPARATUS.

Of the above instruments, the pressure-measuring instruments, the circuit breakers, and the gas sampler and its circuit maker are placed in the various stations in the mine. The tenth of a second time marker,

the two automatic commutators, the chronograph, the half-second clock, the batteries, and the operating switches are placed in the observation room. (See Pl. VI, *B*.)

The connections necessary for operating these instruments are made through two cables which run from the observatory to all the stations in the mine (see fig. 8) through a pipe embedded in the rib and faced with concrete (see Pls. II, *A*, V, *A*, VI, *A*, VII, *A*). The smaller cable contains three wires and supplies electrical power at 110 volts for the manometer motors, the gas sampler, and lights in the mine. The larger cable contains 1 No. 10 wire and 30 No. 16 wires (B. & S. gage) and carries low-voltage current from the storage batteries only. This cable supplies the two shot-firing lines, the tenth of a second and the ignition lines to the manometers, and the circuit breaker lines, besides a number of spare wires for special use. The No. 10 wire is used as a common return to the observation room for all circuits.

DETAILS OF EXPLOSION TESTS.

The most important details of the 15 experimental-mine explosion tests that constituted the major part of the investigation here reported are given below in condensed form.

TEST 1.

Date.—October 24, 1911.

Purpose.—To determine the possibility of obtaining a large coal-dust explosion at the mine and to obtain records of the resulting pressures and velocities.

Point of origin of explosion.—Face of main entry, 722 feet from mouth.

Method of ignition.—Blown-out shot from a 1½-inch pipe in the coal.

Charge.—Two pounds of FFF black blasting powder, tamped with 2½ inches of damp clay.

COAL DUST.

Source.—Dust was ground from bituminous coal from the Pittsburgh bed at Oak mine, near Pittsburgh.

Size.—Over 98 per cent passed through a 100-mesh sieve.

Analysis.—

Moisture.....	1. 94
Volatile matter.....	35. 11
Fixed carbon.....	57. 73
Ash.....	5. 22
	100. 00
Sulphur.....	1. 25

Zone.—Main entry, E 722^a to E 300, or 422 feet; gallery slant: Some old coal dust remaining on side shelves from previous gallery tests.

Position.—E 722 to E 702 on bench just below the hole containing igniting charge; E 712 to E 472 on temporary side shelves; E 472 to E 300 on transverse "book" shelves.

Quantity.—500 pounds of coal dust, or a little more than 1.18 pounds per linear foot of entry. The load was equivalent to 0.315 ounce per cubic foot of space or 315 grams per cubic meter.

Inert zone.—E 300 to opening, no coal-dust load; air course, no coal-dust load.

^a An expression consisting of a number preceded by the letter E refers to a point in the main entry distant from the end of the buttress wall at the mouth of the entry by the number of feet indicated.

VENTILATION AND HUMIDITY READINGS.

Ventilation.—5,000 cubic feet of air moving through last cut-through.

Humidity.—

Station.	Wet bulb.	Dry bulb.	Humidity.
	°F.	°F.	Per cent.
Outside mine:	44	52	52
Third cut-through .	56	57½	91

Barometer.—28.88 inches.

INSTRUMENTS AND RECORDS.

MANOMETERS.

Location.—Manometer A at station E 150; manometer B at station E 50.

Spring.—Thirty-pound springs were used in both manometers.

Records.—See figure 15.

PRESSURE CIRCUIT BREAKERS.

Location.—At stations E 150 and E 50, 100.25 feet apart.

Recorded time between stations.—0.0513 second.

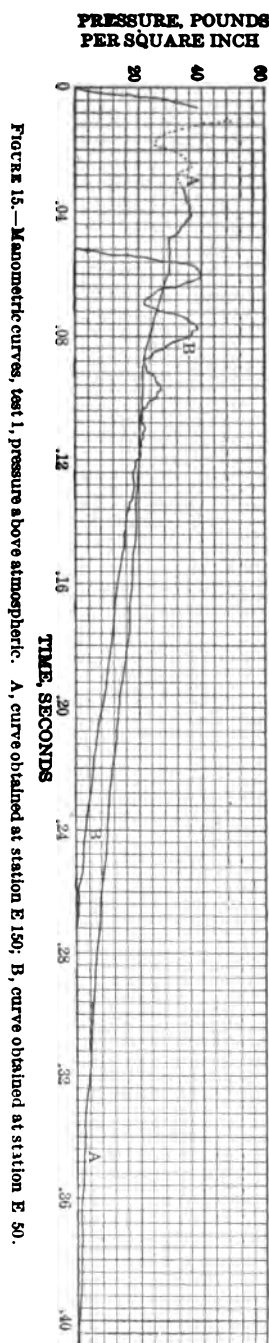
Recorded velocity between stations.—1,954 feet per second.

Adjustment.—Could be operated by a pressure of 5 pounds.

OUTSIDE MANIFESTATIONS.

Length of flame.—Huge masses of flame burst out of the openings. Flame from the main opening estimated to have extended 200 to 300 feet (fig. 16).

Violence of explosion.—The roof of the fan drift was blown off and landed on the tram-road track, 25 feet distant. The 2-inch plank stopping at the end of the steel gallery was blown out. The sandbags on the explosion-door section were thrown high in the air, and the board framework beneath them was destroyed.



A car that stood 50 feet from the opening and was filled with small boards was blown over the side of the slate dump and fell across a wire fence 90 feet from its first position.

A 5-ton gasoline locomotive that had stood just beyond the car was moved 15 feet, and its top plate, pivoting on the rear bolts, was bent through 180°. A car dump which had been at the end of the slate dump was blown 30 feet over the west side of the slate dump.

A steel-plate door weighing about 60 pounds was blown from a point a short distance within the main entry over the dump and was half buried in the ground in the valley beyond the dump, 200 feet from the opening.

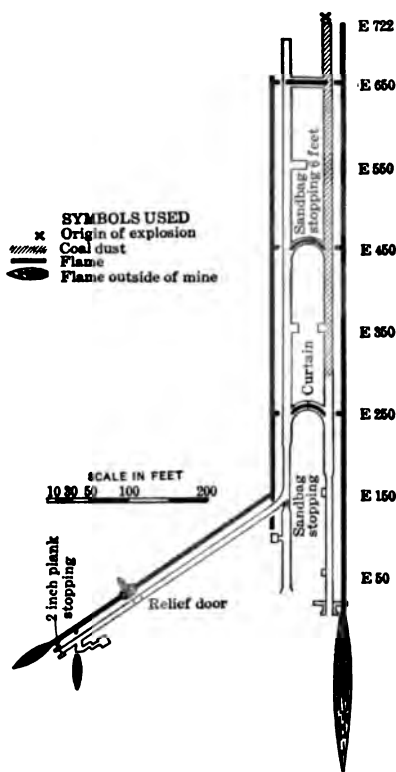


FIGURE 16.—Test 1, extent of flame and conditions of test.

INSIDE OBSERVATIONS.

Violence.—The concrete wall over the top of the steel-plate door at station E 50 was cracked through into the station. The concrete lining was cracked along the springing line on both sides of the entry for 50 feet inby E 50. The concrete arch was cracked from side to side at three points, 18, 30, and 54 feet from the opening. All the steel rails and timbers which had held up the roof at the intersection of the first cut-through and the main entry were thrown down. Some of the timbers were carried out the entry and some through the air course, but the rails had simply dropped out of place. The sandbag stopping in the second cut-through was piled against the inby rib.

Some bags were blown into the entry and some into the air course. All the posts and shelves outby the last or inner cut-through had been blown down with the exception of three posts on the east side of the entry. The stopping in the air course outby the intersection with the gallery slant had been blown down and some of the bags had been carried through the air course to the opening 117 feet distant. The concrete arch in the air course just inby the stopping was cracked from side to side, as was also the arch of the gallery slant at the intersection.

Cracks in the gallery concrete along the springing line extended for 54 feet outby the intersection with the air course.

Coked dust.—Very little coked dust was noted. There was a deposit on the north rib of the first cut-through near the air course on exposures facing the air course. Some coked dust was deposited on the inby side of the cap of the first timber set in the air course outby the first cut-through.

Soot filaments.—In the main entry inby the last cut-through were found many soot filaments hanging from the roof and ledges on the ribs. Some of these were 2 inches in length. A sample of these filaments was analyzed with the following result:

Moisture.....	1.51
Volatile matter.....	27.15
Fixed carbon.....	36.33
Ash.....	35.01
	100.00

[Volatile matter : fixed carbon :: 42.77 : 57.23.]

It is possible that the high ash content was due to impurities having gotten into the sample when the soot filaments were scraped off the slate roof.

NOTES.

It had been planned to cause this explosion by firing two shots, one from the pipe embedded in the face of the entry and another immediately afterwards from a cannon 18 feet from the face. It was thought that the first shot would blow up a cloud of coal dust and the second shot would ignite this cloud. However, the first shot ignited the dust and caused the explosion. The explosion disarranged the firing wires for the cannon and its charge was not fired.

TEST 2.

Date.—October 30, 1911.

Purpose.—Public exhibition of the explosibility of coal dust and a determination of the resulting pressures and velocities.

Witnesses.—Representative mining men from all parts of the United States. Attendance, about 1,500 persons. Over 1,200 persons passed through the mine and inspected the coal-dust distribution and the position of the shot.

Point of origin of explosion.—Face of main entry, E 725. (The face had been advanced 3 feet since the first explosion test.)

Method of ignition.—Four blown-out shots; three from 1½-inch pipes in the coal and one from a 1½-inch pipe lying on the bottom.

Charge.—A total of 9½ pounds of FFF black blasting powder in four holes; each charge tamped with 4 inches of damp clay stemming.

COAL DUST.

Source.—The coal dust was ground from bituminous coal from the Pittsburgh bed at Oak mine, near Pittsburgh.

Size.—Over 98 per cent passed through a 100-mesh sieve.

Analysis.—

Moisture.....	1.94
Volatile matter.....	35.11
Fixed carbon.....	57.73
Ash.....	5.22
	100.00
Sulphur.....	1.25

Zone.—Main entry, 691 feet from face; first cut-through, air course between first cut-through and gallery slant, and gallery slant, 481 feet. Total length of coal-dust zone 1,172 feet.

Position.—E 725 to E 709 on bench below the hole containing the igniting charge; E 712 to E 34 on temporary side shelves, and on 12 "book" shelves placed at intervals; first cut-through on four transverse "book" shelves; air course and slant on temporary side shelves.

Quantity.—Six hundred and twenty-seven pounds in the main entry (average cross section 60 feet), or 0.907 pound per linear foot, which is equivalent to 0.242 ounce per cubic foot (242 grams per cubic meter); 225 pounds in the branch from the entry to the gallery slant (average cross section 48 feet), or 0.468 pound per linear foot, which is equivalent to 0.156 ounce per cubic foot (156 grams per cubic meter).

Inert zones.—No coal-dust loading in the air course inby the first cut-through or outby the stopping at the gallery slant.

VENTILATION AND HUMIDITY READINGS.

Ventilation.—The fan was furnishing to the gallery slant 10,600 cubic feet of air, and 5,000 cubic feet was passing through the last cut-through.

Humidity.—About 11.30 a. m. a drizzling rain began which continued the rest of the day, so that the air throughout the mine was probably in a nearly saturated condition at the time of the explosion.

ATTEMPTS TO FIRE IGNITING SHOT.

After all the visitors who wished to inspect the mine had done so, the shot-firing lines were connected to the lead wires of the electric igniters in the charge, and after all persons had withdrawn to positions of safety, and the prearranged warning signals given, the firing button was pressed, closing the circuit. No explosion resulted. An inspection

of the shot-firing lines in the mine disclosed a short circuit in the lines where they had been trampled by some of the visitors. A second trial resulted in a second failure. As the place of short-circuiting was not found by a second inspection, an entire new shot-firing line was installed. It was evening before the warning signals were given for a third time, when the button was pressed and an explosion resulted.

INSTRUMENTS AND RECORDS.

MANOMETERS.

Location.—Manometer A at station E 150; manometer B at station E 50.

Spring.—Thirty-pound springs were used in both manometers.

Records.—As the motors which drive the drums of the recording manometers did not run during the explosion, the records showed the maximum pressures only. Maximum pressure of manometer A, 26.5 pounds; manometer B, 21.3 pounds.

PRESSURE CIRCUIT BREAKERS.

Pressure circuit breakers were installed at station E 150 and station E 50.

Records.—No velocity records were obtained. When new shot-firing wires were installed after the second failure to obtain the explosion, the batteries used to operate the recording apparatus were disconnected from the apparatus and connected to the shot-firing line.

OUTSIDE MANIFESTATIONS.

LENGTH OF FLAME.

The flame burst out of all openings almost simultaneously and seemed to fill the valley from the main opening to the opposite hill (fig. 17). It was estimated that the flame from the main opening extended horizontally nearly 500 feet, and from 200 to 300 feet vertically. The vertical estimate is probably much too great, the glare on the clouds making the height seem greater than it really was. A tree 153 feet from the main opening was ignited at a point 46 feet above the level of the ground at the mine opening.

INDICATIONS OF VIOLENCE.

Before the explosion a mine car containing several hundred pounds of material had been on the mine track in the line of the entry 40 feet from the opening. Twenty-five feet beyond the car mentioned had stood a car loaded with coal, the total weight of which was

about 2 tons. The partly filled car was hurled over the loaded one and thrown a distance of 184 feet into the ravine beyond the dump. It bounded four times after the first landing and came to rest at a point 45 feet beyond the first landing place; the total travel, measured horizontally, was, therefore, 229 feet. The loaded car first

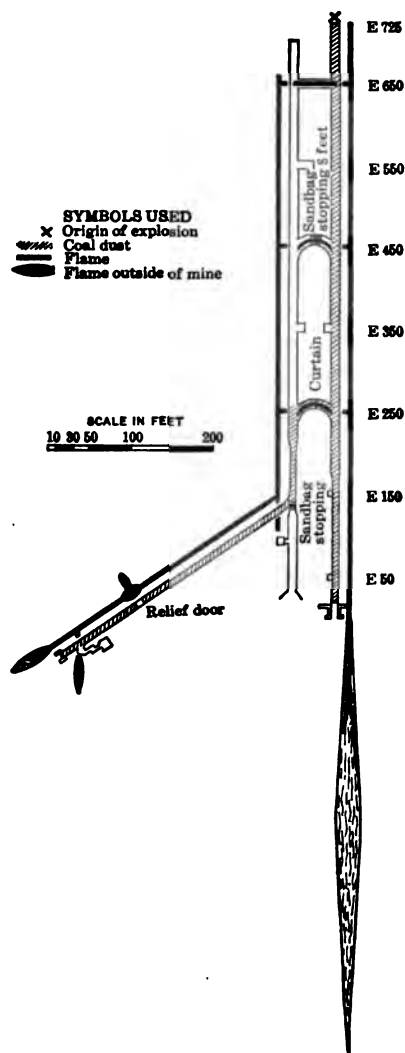


FIGURE 17.—Test 2, extent of flame and conditions of test.

mentioned, even with the brakes set, was driven along the track and finally derailed 70 feet beyond its original position. The hill opposite the mine was strewn with pieces of shelving and posts (Pl. V, B). Posts 5 to 6 feet long and 6 inches in diameter were found 413, 354, and 250 feet from the mine opening.

A sandbag from the sandbag stopping in the air course at the intersection of the air course and gallery slant was found 45 feet from the air-course opening. Many other bags were found between this point and the opening; many of them had been torn and the sand spilled out. The sandbag farthest from the explosion-door section that was blown to the north was found 115 feet distant. Many bags were found scattered between this point and the explosion-door section. A considerable number of the bags were blown high in the air and a rain of sand resulted from their becoming broken and torn. One of the bags was caught in a tree near the explosion-door section at a height of 72 feet above its original position.

The "danger" signboard nailed on the cap at the air-course opening was blown off and was found 140 feet to the northeast. A piece of corrugated iron 2 feet wide and 7 feet long, which had been placed over the bags at the explosion-door section, was found 110 feet to the north. A second sheet of corrugated iron from the explosion-door section was found 260 feet

to the northeast. A telephone pole 5 inches in diameter, which stood northeast of the tool house, was struck by a flying timber from the fan drift and was broken square off at a point 4 feet 9 inches from the ground. A second piece of the fan drift was found 100 feet northeast of its original position. Two windows on the south side of the tool house fell out toward the fan drift; this was no doubt due to a suction following the pressure wave of the explosion. Two lights in a window on the south side of the blacksmith shop were broken. The 1-inch board door and stopping at the end of the steel gallery was blown into small pieces and scattered over the dirt backstop. Flame also issued from the end of this gallery. Many sandbags from the explosion-door section fell to the south of the gallery, some as far as 100 feet up the hill. One bag became caught on a limb of a tree, 42 feet from the ground and 110 feet from the explosion-door section. A piece of corrugated iron was found on the upper side of a fence at a point 110 feet south from its original position. Many pieces of board were found in this field, some as far distant as 235 feet from their positions before the explosion.

INSIDE INDICATIONS OF VIOLENCE.

The action of great force was indicated in the concrete section of the main entry by numerous fractures of the concrete, particularly along the springing lines, and by the exposure and bending of many of the vertical reinforcing rods. The concrete arch had manifestly been lifted by the explosion, thus causing the vertical reinforcing rods to be thrown in tension. The rods tended to become straight where they curved around the arch at the springing line, and this tendency caused the breaking out of the concrete between the rods and the face of the concrete. A crack in the concrete at the springing line on the east side of the entry extended from a point 44 feet from the opening to a point 118 feet inby. On the west side the springing-line crack extended 44 feet to a point 117 feet inby. A large transverse crack (Pl. VI, *C*) was opened in the concrete arch at this point; for some distance inby this point the arch seemed to have been lifted bodily out of its place, and at this point it did not return to its former position but still remained 11 inches higher up. When the concrete lining was originally put in at this point a fall of roof occurred, and after the concrete lining had been put in, loose dirt was tamped into the hole in order to fill it as completely as possible; later the material settled, as there probably was a cavity above the concrete at this point. Practically all of the posts of the coal-dust shelving sets, from the concrete section in to E 612, were blown out. Inby this point the shelving sets seemed to have been merely upset. There had been a 7-foot sandbag stopping in the second cut-through. Practically all of the stopping bags were moved out of place; most of

them were thrown against the inby rib, and some were scattered out into the entry and air course. The inby rib was coated with a layer of sand from the broken bags.

The center hole at the face was enlarged or "cratered" for several feet. The west shot merely "cratered" the front of the hole slightly, while the east shot cracked the coal vertically. The hole from the center shot was partly filled with dirt and broken pieces of wood which seemed to have been blown back into the hole by an air movement rushing back toward the face to fill the vacuum following the pressure wave of the explosion.

There had been a sandbag stopping across the air course just outby its intersection with the gallery slant. This was thrown down and the bags scattered along the air course to a point 55 feet outside of the opening. After the explosion the intrushing air carried inby some of the sandbags to the intersection with the gallery slant and some were thrown by the blast into the gallery slant to points 15 to 54 feet from the intersection.

The concrete pillar at the intersection of the two entries was cracked horizontally about $3\frac{1}{2}$ feet from the bottom and while the upper half was raised by the pressure of the explosion, a piece of brattice cloth and a stick were caught in the crack when the concrete descended (Pl. VIII, A). The wooden doors in the air course were blown open and the hinge posts moved outby 3 to 4 inches. The lifting of the arch of the concrete section of the gallery slant caused longitudinal cracks along the springing lines almost the entire length of the entry. There were numerous exposures of the vertical reinforcement near the springing line, and in some cases the short diagonal braces from the crossbars to the posts of the temporary timber sets behind the concrete lining were exposed.

At G 168^a one of the vertical reinforcement rods was exposed for $3\frac{1}{2}$ feet and the middle portion was bent 7 inches outby the original position. At G 164^b the roof in rising pulled the reinforcing rod up and when it settled again the rod became buckled (Pl. VIII, B); measurements of the buckled portion indicated that the roof had lifted 1 foot. The cover at this point was 8 feet. At G 161^c the concrete was cracked across the arch. The roof seemed to have settled slightly inby its original position.

COKE AND COKED DUST.

There were coke deposits on the inby corner of the entry and second cut-through and some coked dust on the west rib opposite

^a One hundred and sixty-eight feet from outer end of iron gallery or 26 feet from mouth of gallery slant.

^b One hundred and sixty-four feet from outer end of iron gallery or 22 feet from mouth of gallery slant.

^c One hundred and sixty-one feet from outer end of iron gallery or 19 feet from mouth of gallery slant.

the second cut-through. At E 560 there were coked-dust deposits on the outby exposures of both the east and the west ribs, also in the inby cracks on the entry rib near the third cut-through. At E 635 on a shelving post still standing there was a deposit of coked dust on the inby side.

In the third cut-through the coal on each rib was blistered very badly (Pl. X). Coked-dust deposits began to appear about 20 feet from the entry in going toward the air course on the inby side of the cut-through on the exposures facing the air course and increased in amount toward the air course. Practically all coked dust on the outby side of the cut-through was on the exposures facing the entry.

At A 654 ^a there were heavy deposits of coked dust in the crevices at the corner of the cut-through.

Twenty feet inby the cut-through on the west side of the air course were deposits of coked dust on the outby exposures and coke bubbles along the bedding planes of the coal (Pl. XI). This coking effect has been termed "coke in situ."

There were frequent deposits of coked dust (Pl. XII, A) on both sides of the air course from the third cut-through to the second cut-through, principally on the outby exposures. From the second cut-through to the first the deposits were smaller and less frequent.

On the north rib of the blind cut-through at A 550 near the corner of the air course the rib was considerably blistered, and particles of coke were formed in place, grouped along the bedding planes of the coal bed.

SOOT COATING.

In the entry inby the third cut-through, the walls, roof, and floor were coated with soot, and threads or filaments of soot up to several inches in length were hanging from the roof (Pl. XII, B). The unburned hydrocarbons were seemingly deposited from the quiet atmosphere after the explosion, the ventilation not being reestablished until the following day. The analysis of the hanging filaments was as follows:

Moisture.....	2.25
Volatile matter.....	41.48
Fixed carbon.....	49.60
Ash.....	6.67
	100.00

[Volatile matter : fixed carbon :: 45.54 : 54.46.]

The ash content was slightly higher than the ash content of the original dust, but the percentage of volatile matter in the ratio of volatile matter to fixed carbon increased considerably.

^a A point in the air course nearly opposite E 644.

ANALYSES OF SAMPLES.

Samples of coked dust, blistered coal, stalactitic carbon, and road dust were obtained and analyzed after the explosion.

The results of a proximate analysis of a combined sample of blistered coal from the west rib of the air course and the north rib of the last cut-through are given below. For purposes of comparison the results of an analysis of a representative sample of the unaltered coal are given also.

Results of analyses of blistered and of unaltered coal.

Constituent.	Blistered coal.	Unaltered coal.
	<i>Per cent.</i>	<i>Per cent.</i>
Moisture.....	1.64	2.73
Volatile matter.....	34.56	36.03
Fixed carbon.....	60.49	54.98
Ash.....	3.31	6.26
Total.....	100.00	100.00

[Proportion for blistered coal—Volatile matter : fixed carbon :: 36.36 : 63.64; proportion for unaltered coal—Volatile matter : fixed carbon :: 39.59 : 60.41.]

A comparison shows that the percentage of volatile matter as compared with fixed carbon is less in the blistered coal than in the unaltered coal. The low ash content of the blistered coal undoubtedly is not significant of change. By accident the layer of coal had a much lower content than normal.

Two other samples of blistered coal were taken in the third cut-through. One was taken by cutting off as thin a slice as possible from the blister; the other represented a second thin layer beneath the first. It was thought that the analysis of the first sample, because of the blistering that caused the curved surfaces on the face of the coal, might show different results than did the analysis of the unaltered coal. The results of both analyses were as follows:

Results of analyses of two layers of blistered coal.

Constituent.	First layer.	Second layer.
Moisture.....	1.82	1.73
Volatile matter.....	36.17	37.83
Fixed carbon.....	55.45	57.25
Ash.....	6.56	3.17
Total.....	100.00	100.00

[Proportion for first layer—Volatile matter : fixed carbon :: 39.48 : 60.52; proportion for second layer—Volatile matter : fixed carbon :: 39.79 : 60.21.]

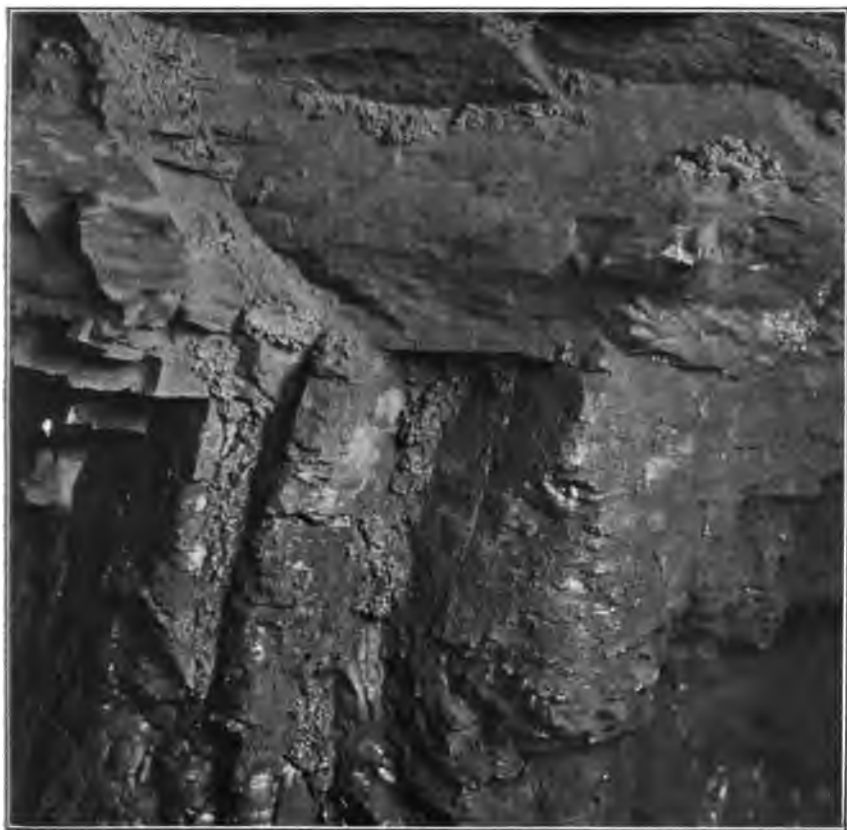


BLISTERED COAL RIB, TEST 2.

Note rounded faces where the outer shell of soot-coated coal spalled off after cooling.



COKE BUBBLES FORMED ALONG BEDDING PLANES OF COAL TEST 2.



A. COKED DUST, TEST 2.



B. COATING AND FILAMENTS OF SOOT, TEST 2.

The results of the analyses show practically no change in composition as compared with the results of analysis of unaltered coal. The low percentage of ash shown by the second analysis is not significant on account of the small size of the sample.

The bright conchoidal surfaces of a blister were exposed when a thin scale dropped from the blister. As both the scale and the blistered coal were included in the above analyses, the results would indicate that a change in physical appearance does not seem to be accompanied by any great change in chemical composition.

The results of an analysis of the standard dust used in this experiment were as follows:

Moisture.....	1. 94
Volatile matter.....	35. 11
Fixed carbon.....	57. 73
Ash.....	5. 22
	100. 00

[Volatile matter : fixed carbon :: 37.82 : 62.18.]

A sample of coked dust from the outby exposures on the inby corner of the last cut-through and the air course was analyzed with results as follows:

Moisture.....	1. 51
Volatile matter.....	22. 32
Fixed carbon.....	42. 39
Ash.....	33. 78
	100. 00

[Volatile matter : fixed carbon :: 34.49 : 65.51.]

This analysis shows a decrease of the percentage of volatile matter in the ratio of volatile matter to fixed carbon, and a large increase in the percentage of ash, probably caused by partial combustion of the coal dust and to a mixing of the coal dust with some high-ash material from the roof or floor.

Analysis of a sample of the road dust showed the following results:

Moisture.....	2. 00
Volatile matter.....	24. 22
Fixed carbon.....	33. 01
Ash.....	40. 77
	100. 00

[Volatile matter : fixed carbon :: 42.32 : 57.68.]

This analysis shows a large increase in the percentage of ash, as might be expected, because the sample was taken from the entry bottom where the coal dust might have become mixed with incom-bustible material and because a large proportion of the combustible material in the coal dust had been consumed. In regard to the increased proportion of volatile matter to fixed carbon the possible explanation is that the sample contained a quantity of highly volatile

soot which settled out of the smoke after the explosion, rather than that more fixed carbon than volatile matter had been burned in the explosion, but may be due to the water of combination of the included shale or limestone dust.

TEST 3.

Date.—January 31, 1912.

Purpose of test.—Study of the propagation of coal-dust explosions and the effect of a Taffanel stone-dust barrier.

Special conditions.—Subsequent to October 30, 1911, permanent shelving had been installed throughout the length of the main entry; 5 new circuit-breaker stations had been installed at points E 250, E 350, E 450, E 550, and E 650; and a reinforced-concrete stopping had been built in the first cut-through.

Point of origin of explosion.—Face of main entry, E 738.

Method of ignition.—Blow-out shot from a 1½-inch pipe in the coal.

Ignition charge.—Two pounds of FFF black blasting powder, tamped with 5 inches of damp clay stemming.

COAL DUST.

Source.—The coal dust was ground from bituminous coal from the Pittsburgh bed at Willock, Pa.

Size.—Over 95 per cent passed through a 100-mesh screen.

Analysis.—

Moisture.....	3.13
Volatile matter.....	32.71
Fixed carbon.....	54.40
Ash.....	9.76
	100.00
Sulphur.....	1.44

Zone.—Main entry, E 738 to E 654, or 84 feet.

Position.—E 738^a to E 722 on a bench placed just below the level of the hole containing the igniting charge; E 733 to E 654 on permanent side shelves on both sides of the entry.

Quantity.—One hundred and five pounds of coal dust: 26 pounds on bench and remainder on side shelves, distributed 1 pound per foot.

Inert zone.—The main entry was watered from E 650 to the opening. The air course was also sprinkled throughout its length.

^a The main entry face was advanced from E 725 to E 738 in the interim between tests 2 and 3.

TAFFANEL BARRIER.

Location.—In the air course, extending from 5 to 85 feet outby third cut-through.

Number of shelves.—Eleven.

Distance between shelves.—About 8 feet from center to center.

Width of shelves.—Twenty-four inches.

Depth of shelves.—Two inches.

Depth of stone dust.—Seven and one-half to eight inches.

Area of obstruction.—13.81 per cent of total cross section.

Analysis of stone dust.—

Organic hydrogen.....	0.15
Organic carbon.....	1.98
Moisture (at 105° C.).....	2.57
Inorganic volatile matter (combined water, CO ₂ , etc.).....	8.42
Ash.....	86.88
	100.00

Ventilation and humidity readings.

Station.	Wet bulb.	Dry bulb.	Barometer.	Humidity.	Cross section.	Velocity.	Volume.
	° F.	° F.	Inches.	Per cent.	Feet.	Feet per minute.	Cubic feet per minute.
G 310*.....	26	29	28.89	67	6½ by 6½	192	7,488
E 650.....	43	45	28.85	86			
E 50.....	41	42	28.875	92	9 by 6½	75	4,388

* 310 feet from mouth of gallery.

INSTRUMENTS AND RECORDS.

MANOMETERS.

Location.—Manometer A (suction) at station E 50; manometer B (pressure) at station E 50; manometer C (pressure) at station E 150.

Springs.—Manometers B and C were equipped with 200-pound springs.

Records.—Pressure too small to be recorded.

PRESSURE CIRCUIT BREAKERS.

Location.—At stations E 50, E 150, E 250, E 350, E 450, E 550, E 650.

Adjustment.—Set to operate at a pressure of 5 pounds.

Records.—Pressure circuit breaker at station E 650 operated.

FLAME CIRCUIT BREAKERS.

Type.—Detonator, coated inside and outside with silver carbide (see p. 63).

Location.—Stations E 50 to E 650, inclusive.

Records.—Flame circuit breaker at station E 650 operated.

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OUTSIDE MANIFESTATIONS.

A cloud of dust blew out the explosion-door section first. The doors at the top of the shaft blew open and a cloud of light-colored dust came out. At the main entry a little dust blew out the opening. The most noticeable feature outside the mine was the swirling up of the snow from the main opening to the end of the dump.

INSIDE OBSERVATIONS.

Few indications of violence were noted. The first door in the air course was blown open 2 feet. A board from the door was found in

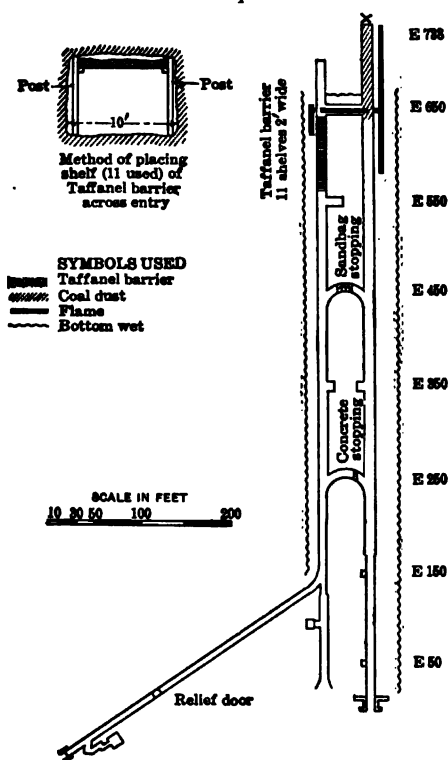


FIGURE 18.—Test 3, extent of flame and conditions of test.

the air course 6 feet outby. The second door was cracked in the middle and the iron straps on it were bent inby. There was more or less coal dust scattered throughout the air course, there being heavy deposits of coal dust on the shelves of the Taffanel barrier.

Black damp.—The doors were closed so that the black damp could be cleared out. Three minutes after the doors had been closed a heavy cloud of smoke started to come out and continued for 15 minutes.

Length of flame.—The length of the flame was indicated by guncotton tufts that had been suspended from the roof at intervals of 25 feet. The flame in the entry was 159 feet long, and the branch flame in the third cut-through and in the air course was 63 feet long (fig. 18).

Efficiency of barrier.—It is to be noted that the flame did not extend through the barrier. The flame might or might not have extended farther if the barrier had not been there. The barrier probably did not have much of a quenching effect as the stone dust was only slightly disturbed.

TEST 4.

Date.—January 31, 1912.

Purpose.—To determine the possibility of an explosion from the same dust as was used in preceding test.

Point of origin of explosion.—Face of main entry, E 738.

Method of ignition.—Blown-out shot from 1½-inch pipe in the coal. The hole in which the igniting charge was placed was 2 feet 4 inches above the bottom, and 16 inches from the west rib. The axis of the hole intersected the floor 16 feet from the face.

Ignition charge.—Two pounds of FFF black blasting powder, tamped with 5 inches of damp clay stemming.

COAL DUST.

Zone.—E 738 to E 654, 84 feet.

Position.—E 738 to E 732 on bench placed just below the level of the hole containing the igniting charge; E 733 to E 654 on permanent side shelves on both sides of the entry.

Quantity.—Twenty-six pounds of fresh dust was placed on bench in front of shot; dust on the shelves was that remaining from the preceding test in which 105 pounds had been used.

Inert zone.—Preceding the previous test the main entry had been watered from E 650 to the opening. Some coal dust had been blown from the shelves onto the floor of the entry from E 650 outby, so that this zone was not as inert as it had been in test 3. The air course, for a like reason, had a thin coating of coal dust on the floor.

TAFFANEL BARRIER.

Location.—In the air course, extending from 5 to 85 feet outby the third cut-through.

Number of shelves.—Eleven.

Distance between shelves.—About 8 feet from center to center.

Width of shelves.—Twenty-four inches.

Depth of shelves.—Two inches.

Depth of stone dust.—Seven and one-half to 8 inches.

Obstruction.—13.81 per cent of the total cross section.

Analysis of stone dust.—

Organic hydrogen.....	0. 15
Organic carbon.....	1. 98
Moisture (at 105° C.).....	2. 57
Inorganic volatile matter (combined water, CO ₂ , etc.).....	8. 42
Ash.....	86. 88
	100. 00

INSTRUMENTS AND RECORDS.

No instruments were used in this test.

OUTSIDE MANIFESTATIONS.

The steel sheets over the explosion-door section rattled, but were not moved to any great extent by the explosion. The second puff blew open the east door on top of the air shaft. Eight well-defined puffs of smoke were seen to come from the shaft. The only indication of an explosion at the main entry was a flurry of snow and dust from the opening to the end of the dump.

INSIDE OBSERVATIONS.

An examination of the face of the entry showed that the pipe in which the charge had been placed was split its entire length, and the coal was broken for 18 inches back from the front of the hole.

Shelves from E 650 to E 350 had a fine coating of coal dust which had been blown from the dust zone. A deposit of coked dust was found on the east shelf outby the third cut-through.

The shelves from E 738 to E 610 were coated with soot deposited from the smoke of the explosion. A small deposit of coked dust was found on the shelves inby E 670.

LENGTH OF FLAME.

None of the strips of guncotton unaffected by the preceding explosion was affected by this explosion. It is probable that the length of flame of the two explosions was about the same.

TEST 5.

Date.—February 2, 1912.

Purpose.—In test 3 of January 31, the circuit breaker at station E 650 operated, but none of the other circuit breakers was acted on by sufficient pressure to break the circuits. It was thought that had the circuit breakers in the preceding explosion been so adjusted that they would have been operated by less pressure, more records would have been obtained. Therefore, this test was made to determine what was the least pressure to which they could be adjusted and still not be operated by a blown-out shot in the coal face.

Point of origin of explosion.—Face of main entry, E 744.

Method of ignition.—Blown-out shot from 1½-inch pipe in the coal.

Charge.—Two pounds of FFF black blasting powder tamped with 5 inches of damp shale dust.

Coal dust.—No coal dust used in this test.

Inert zone.—Main entry watered the entire distance.

Ventilation.—In order to determine whether the mine was producing any gas, the fan was shut down for 14 hours preceding this test and two samples of gas were taken; one at the face of the air course and one at the face of the entry, with results as follows:

Analysis of air-course sample.

CO ₂	0.07
O ₂	20.59
CO.....	.00
CH ₄	.06
N.....	79.28
	100.00

Analysis of entry sample.

CO ₂	0.06
O ₂	20.58
CO.....	.00
CH ₄	.04
N.....	79.32
	100.00

The above results of analyses show that very little methane had accumulated during the time the fan had been shut down. On the basis of the analysis results no perceptible amount of gas would be found with the ventilation in normal condition. After the samples had been taken the fan was started so that the ventilation conditions of this test would be similar to those of test 3.

INSTRUMENTS AND RESULTS.

Pressure circuit breakers.—Two pressure circuit breakers were installed in the station at E 650; one was adjusted so as to be operated by a pressure of 1 pound and the other by a pressure of 2 pounds.

Length of flame.—Guncotton tufts at E 704 and E 729 were ignited either by the flame from the shot or more probably by a flying particle of burning powder.

Results.—Only the circuit breaker set at 1 pound operated. It was concluded, therefore, that circuit breakers set at 2 pounds would not be operated by the pressure from a blow-out shot from a 1½-inch pipe, with the charge and the stemming of this test.

TEST 6.

Date.—February 3, 1912.

Purpose.—Study of the propagation of a coal-dust explosion, and of the effect of Taffanel stone-dust barrier with coal-dust zone of 190 feet.

Origin of explosion.—Face of the main entry, E 744.

Method of ignition.—Blown-out shot from 1½-inch pipe in the coal.

Charge.—Two pounds of FFF black blasting powder, tamped with 5 inches of damp shale dust.

COAL DUST.

Source.—The coal dust was ground from bituminous coal from the Pittsburgh bed at Willock, Pa.

Size.—Over 95 per cent passed through a 100-mesh screen.

Analysis.—

Moisture.....	3.13
Volatile matter.....	32.71
Fixed carbon.....	54.40
Ash.....	9.76
	100.00
Sulphur.....	1.44

Zone.—E 744 to E 554, or 190 feet.

Position.—E 744 to E 728, on bench placed just below the level of the hole containing the ignition charge; E 733 to E 554, on permanent side shelves on both sides of the entry.

Quantity.—Twenty-five pounds of coal dust on the bench; 175 pounds on the side shelves; loading equivalent to 1.05 pounds per linear foot of entry.

Inert zone.—Main entry, from E 554 outby, was sprinkled. Air course also was sprinkled before the explosion.

TAFFANEL BARRIER.

Location.—In the air course, extending from 5 to 85 feet outby third cut-through.

Number of shelves.—Eleven.

Distance between shelves.—About 8 feet from center to center.

Width of shelves.—Twenty-four inches.

Depth of shelves.—Two inches.

Depth of stone dust.—Seven and one-half to eight inches.

Obstruction.—13.81 per cent of total cross section.

Analysis of stone dust.—The coal dust that had settled on the surface of the stone dust after explosion tests 3 and 4 was scraped off, and ground shale dust was heaped on the shelves until they would hold no more. A sample of the dust was taken from various parts of a carload of shale. Its analysis follows:

Organic hydrogen.....	0.39
Organic carbon.....	7.59
Moisture (at 105° C.).....	3.94
Inorganic volatile matter (combined water, CO ₂ , etc.).....	5.42
Ash.....	82.66
	100.00

VENTILATION.

Air and humidity readings taken at two points just prior to the explosion were as follows:

Air and humidity readings, test 6.

Station.	Wet bulb.	Dry bulb.	Barometer.	Humidity.	Cross section.	Velocity.	Volume.
	<i>° F.</i>	<i>° F.</i>	<i>Inches.</i>	<i>Per cent.</i>	<i>Feet.</i>	<i>Feet per minute.</i>	<i>Cubic feet per minute.</i>
E 180.....	45	46½	28.96	89	6½ by 9	80	4,680
Third cut-through.....	39	40	28.96	92	6½ by 9	60	3,330

Analysis of mine air.—A sample of mine air (No. 2078) was taken at the face of the main entry just before firing the shot in order to determine whether any methane was present. The results of analysis were as follows:

CO ₂	0.06
O.....	19.86
CO.....	.00
CH ₄	.02
N.....	80.06
	100.00

The analysis shows 0.02 per cent of methane, and as the possible error would not exceed 0.01 per cent, one may be assured that there was only a slight trace of gas present, too little to affect the result of the shot. It is also probable that even a trace could not be found near the cut-through.

INSTRUMENTS AND RECORDS.

MANOMETERS.

Location.—Manometer A (suction) at station E 50; manometer B (pressure) at station E 50; manometer C (pressure) at station E 150.

Springs.—Manometers B and C were equipped with 200-pound springs.

PRESSURE CIRCUIT BREAKERS.

Location.—Stations E 50, E 150, E 250, E 350, E 450, E 550, E 650.

Adjustment.—Set to operate at a pressure of 2 pounds.

FLAME CIRCUIT BREAKERS.

Location.—Stations E 50, E 150, E 250, E 350, E 450, E 550, E 650.

RECORDS.

Manometers.—Pressure too small to be recorded (less than 1 pound per square inch).

Pressure circuit breakers.—Operated at station E 650.

Flame circuit breakers.—Operated at stations E 650 and E 550.

Velocity of flame propagation.—The flame reached station E 650, 1.8714 seconds after ignition and station E 550, 2.6617 seconds after ignition; the time between those stations was 0.7903 second, and the velocity 127 feet per second.

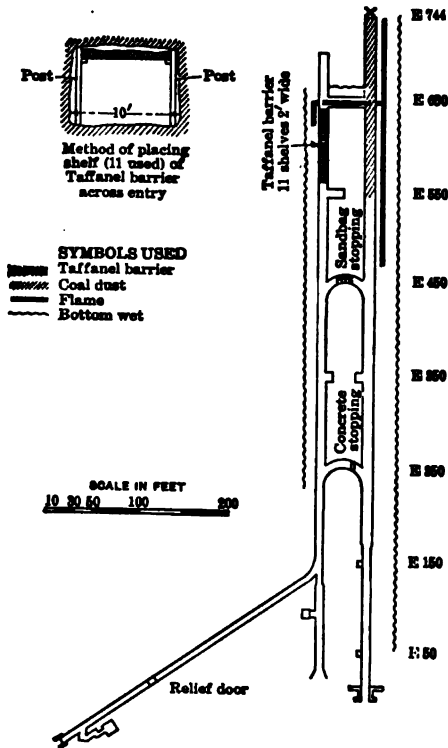


FIGURE 19.—Test 6, extent of flame and conditions of test.

tufts suspended along the entry to be 265 feet long (fig. 19). The branch flame in the cut-through and air course was 72 feet long.

EFFICIENCY OF BARRIER.

The flame extended farther into the barrier than it did in test 3, in which the length of the coal-dust loading in the main entry was 100 feet less than in this experiment. The stone dust was not disturbed very much by the explosion, so that its effect in quenching the flame was probably not great. A sample of the coal dust left on the shelves

OUTSIDE MANIFESTATIONS.

Violence.—The steel sheets of the explosion-door section were rattled by the explosion, and some of them were moved slightly toward the steel gallery. The only other outside evidence of an explosion was the swirl of snow and dust from the opening to the end of the dump.

INSIDE OBSERVATIONS.

The curtain hanging across the air course in by the shaft was carried in by 26 yards. The board door out by the shaft was broken outward.

Length of flame.—The length of flame in the main entry was indicated by the guncotton

after the explosion was taken, the analysis of which gave the following results:

Moisture	2. 85
Volatile matter	33. 02
Fixed carbon	54. 31
Ash	9. 82
	100. 00
Sulphur	1. 24

A comparison of the ratio of volatile matter to fixed carbon (37.81:62.19) with the ratio of volatile matter to fixed carbon in the original coal dust (37.55:62.45) shows that the percentage of volatile matter was not materially changed.

Of the 200 pounds of coal dust used in this test, 76 pounds remained on the shelves after the explosion.

TEST 7.

Date.—February 5, 1912.

Purpose.—Study of the propagation of a coal-dust explosion and the effect of Taffanel barrier with a coal-dust zone of 290 feet.

Point of origin of explosion.—Face of main entry, E 744.

Method of ignition.—Blown-out shot from 1½-inch pipe in the coal.

Charge.—Two pounds of FFF black blasting powder, tamped with 5 inches of damp clay.

COAL DUST.

Source.—The coal dust was ground from bituminous coal from the Pittsburgh bed at Willock, Pa.

Size.—Over 95 per cent passed through a 100-mesh screen.

Analysis.—

Moisture	3. 13
Volatile matter	32. 71
Fixed carbon	54. 40
Ash	9. 76
	100. 00
Sulphur	1. 44

Zone.—From E 744 to E 454, or 290 feet.

Position.—From E 744 to E 728 on a bench placed just below the level of the hole containing the igniting charge; from E 733 to E 454 on permanent shelves along both sides of the entry.

Quantity.—Twenty-six pounds of coal dust was placed on the bench and 324 pounds was along the side shelves. This is equivalent to an average loading of 1.21 pounds per linear foot.

Inert zone.—The main entry was watered from E 450 to the opening; the air course was also sprinkled to render it inert.

TAFFANEL BARRIERS.

Location.—In the air course, extending from 5 to 85 feet outby cut-through.

Number of shelves.—Eleven.

Distance between shelves.—About 8 feet from center to center.

Width of shelves.—Twenty-four inches.

Depth of shelves.—Two inches.

Depth of stone dust.—Seven and one-half to eight inches.

Obstructions.—13.81 per cent of total cross section.

Analysis of stone dust.—

Organic hydrogen.....	0. 39
Organic carbon.....	7. 59
Moisture (at 105° C.).....	3. 94
Inorganic volatile matter (combined water, CO ₂ , etc.).....	5. 42
Ash.....	82. 66
	100. 00

Humidity readings.

Station.	Wet bulb.	Dry bulb.	Humidity.
	° F.	° F.	Per cent.
Outside.....	12	14	63
E 250.....	45	45	100
E 450.....	45	46	93

INSTRUMENTS AND RECORDS.

MANOMETERS.

Location.—Manometer A (suction) at station E 50; manometer B (pressure) at station E 50; manometer C (pressure) at station E 150.

Springs.—Manometers B and C were equipped with 200-pound springs.

PRESSURE CIRCUIT BREAKERS.

Location.—Stations E 50, E 150, E 250, E 350, E 450, E 550, E 650.

Adjustment.—Set to operate at a pressure of 2 pounds.

FLAME CIRCUIT BREAKERS.

Type.—Detonator with silver-carbide stick (p. 63).

Location.—Stations E 50, E 150, E 250, E 350, E 450, E 550, E 650.

RECORDS.

Manometers.—Pressures were too small to be recorded.

Velocity.—No pressure circuit breakers operated. Flame circuit breakers at stations E 650 and E 550 operated. The flame reached

station E 650, 1.4917 seconds after ignition and station E 550, 2.1149 seconds after ignition. The time between stations was 0.6232 second, and the velocity 160 feet per second.

MANIFESTATIONS.

Outside.—The only evidence of an explosion outside was the rattling of the steel sheets of the explosion-door section, and the appearance of a slight cloud of dust at the mouth of the main opening.

Inside.—The door in the air course had been previously removed and the curtains replacing it were not torn down by the explosion. There were no signs of violence in any part of the mine.

LENGTH OF FLAME.

The flame in the main entry extended from the face to E 279, or 465 feet (fig. 20). In the cut-through air-course branch the flame extended 135 feet or to the outby end of the Taffanel barrier. The stone dust on the barrier was disturbed only slightly and it is uncertain just how much of a quenching effect the barrier may have had on the flame.

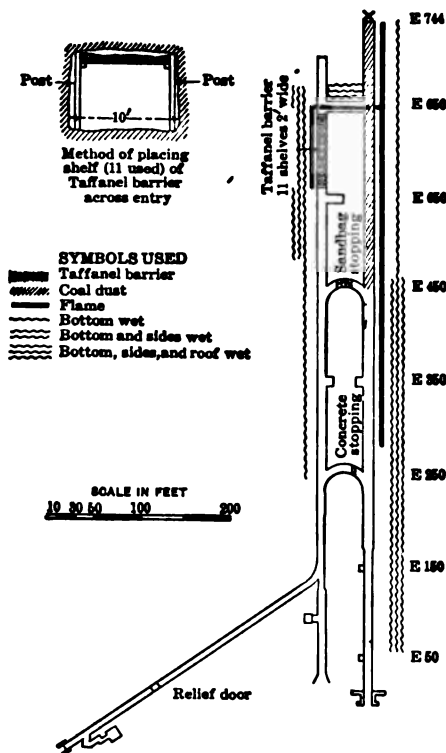


FIGURE 23.—Test 7, extent of flame and conditions of test.

ANALYSES MADE AFTER THE EXPLOSION.

COAL DUST.

A coal-dust sample (No. 13297) was taken from the dust remaining on the shelves after this explosion and was analyzed, the results being as follows:

Moisture.....	3.01
Volatile matter.....	32.64
Fixed carbon.....	50.57
Ash.....	13.78
	100.00
Sulphur.....	1.18

Comparison of the ratio of volatile matter to fixed carbon (39.23:60.77) with the ratio of volatile matter to fixed carbon in the original dust (37.55:62.45) indicates again an increase in volatile matter and a corresponding decrease in the fixed carbon. It is probable that this change was again due to a mixing of the original dust with some soot from the explosion.

One hundred and thirty pounds out of the 350 pounds put on the shelves for this test remained after the explosion.

ROAD DUST.

A road-dust sample (No. 13295) from the air course between the second and third cut-throughs was taken after the explosion, and was analyzed, with the following results:

Moisture.....	16.02
Volatile matter.....	17.10
Fixed carbon.....	13.91
Ash.....	52.97
	100.00
Sulphur.....	.84

The high content of ash and moisture would render this road dust nonexplosive, and might possibly even have a quenching effect on the flame of the explosion.

MINE AIR.

A sample of mine air (No. 2079) was taken at the face of the main entry as soon after the explosion as it was possible to reach this point without a helmet. The analysis showed the following results:

CO ₂	0.04
O ₂	20.93
CO.....	.00
CH ₄	Trace.
N.....	79.03
	100.00

This sample showed a surprisingly small percentage of black damp and no carbon monoxide; the smell and the presence of smoke indicated worse conditions than were shown by this sample.

A sample of mine air (No. 2081) was also taken in by station E 650 about the same time as the preceding sample, its analysis showing the following results:

CO ₂	0.08
O ₂	20.91
CO.....	.00
CH ₄	Trace.
N.....	79.01
	100.00

This analysis is almost a duplicate of that of the preceding sample.

TEST 8.

Date.—February 7, 1912.

Purpose of test.—Study of propagation of a coal-dust explosion and the effect of Taffanel barrier with a coal-dust zone of 290 feet.

Point of origin of explosion.—Face of main entry, E 744.

Method of ignition.—Blown-out shot from a cannon placed near the face.

Charge.—Two and three-fourths pounds of FFF black blasting powder tamped with 3 inches of damp shale dust.

COAL DUST.

The coal dust was ground from bituminous coal from the Pittsburgh bed at Willock, Pa.

Size.—Over 95 per cent passed through a 100-mesh screen.

Analysis.—

Moisture.....	2.94
Volatile matter.....	33.98
Fixed carbon.....	54.36
Ash.....	8.72
	100.00
Sulphur.....	1.32

The sample analyzed was taken from the dust on the shelves just preceding the explosion.

Zone.—From the face at E 744 to E 454, or 290 feet.

Position.—E 742 to E 726 on a bench placed just below the level of the bore hole of the cannon; E 733 to E 454 on permanent side shelves along both sides of the entry.

Quantity.—Thirty pounds of coal dust on the bench and 270 pounds scattered along the shelves. This is equivalent to an average loading of 1.03 pounds per linear foot.

Inert zone.—The main entry from E 450 outby to the opening, the last cut-through, and the air course were watered to render them inert.

TAFFANEL BARRIER.

Location.—In the air course, extending from 5 to 85 feet outby third cut-through.

Number of shelves.—Eleven.

Distance between shelves.—About 8 feet from center to center.

Width of shelves.—Twenty-four inches.

Depth of shelves.—Two inches.

Depth of stone dust.—Seven and one-half to eight inches.

Obstructions.—13.81 per cent of total cross section.

Shale dust.—A sample of the ground shale and limestone on the Taffanel-barrier shelves before the explosion was analyzed, with the following results:

Organic hydrogen.....	0.41
Organic carbon.....	7.48
Moisture (at 105° C.).....	4.28
Inorganic volatile matter (combined water, CO ₂ , etc.).....	7.25
Ash.....	80.58
	100.00

VENTILATION AND HUMIDITY READINGS.

Ventilation.—The fan stopped two minutes before the explosion, so that the explosion probably occurred in still air.

Humidity.—

Station.	Wet bulb.	Dry bulb.	Humidity.
	°F.	°F.	Per cent.
Third cut-through.....	36½	38½	83
E 450.....	40½	42½	85
Outside.....	18	20	70

INSTRUMENTS AND RECORDS.

MANOMETERS.

Location.—Manometer A (suction) at station E 50; manometer B (pressure) at station E 50; manometer C (pressure) at station E 150.

Springs.—Manometers B and C were equipped with 200-pound springs.

PRESSURE CIRCUIT BREAKERS.

Location.—At stations E 50, E 150, E 250, E 350, E 450, E 550, E 650.

Adjustment.—Set to operate at a pressure of 2 pounds.

FLAME CIRCUIT BREAKERS.

Type.—Detonator type with silver-carbide stick.

Location.—At stations E 50, E 150, E 250, E 350, E 450, E 550, E 650.

RECORDS.

Manometers.—The pressure was too low to operate manometers B and C. The record obtained on suction manometer A is reproduced in figure 21.

Velocity.—Pressure circuit breakers at stations E 650, E 550, and E 450 operated.

Flame circuit breakers from station E 650 to station E 250, inclusive, operated.

Velocity of propagation of explosion, test 8.

Station.	Pressure wave.			Flame.		
	Total time from ignition.	Time between stations.	Velocity between stations.	Total time from ignition.	Time between stations.	Velocity between stations.
	<i>Seconds.</i>	<i>Seconds.</i>	<i>Feet per second.</i>	<i>Seconds.</i>	<i>Seconds.</i>	<i>Feet per second.</i>
E 650	0.0685			0.6216		
E 550	.1565	0.0880	1,136	1.0771	0.4555	220
E 450	.2461	.0896	1,116	1.4715	.3944	254
E 350				1.7226	.2511	398
E 250				3.6857	1.9631	51

MANIFESTATIONS.

Outside.—The only outside evidence of an explosion was a rattling of steel sheets over the explosion-door section, and a cloud of dust and smoke and flurries of snow swirling from the main opening to the end of the dump. The brace on the west side of the car dump,

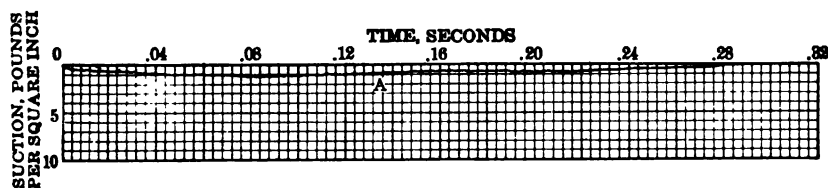


FIGURE 21.—Manometric curve, test 8, pressure below atmospheric.

which was standing at the edge of the slate dump, was broken by a flying board from a concrete form that had been resting against the concrete wall at the opening. Other pieces of the same concrete form were sucked into the entry to station E 50. The top bar of the entry gates was broken into two pieces; one piece was found on the dump 120 feet from the opening, and the other on the east side of the slate dump. One part of the board "Danger" sign on the air course was blown 54 feet and the other 72 feet in from the opening. The last four sheets on the explosion-door section were lifted completely out of place, and fell on the movable section of the steel gallery.

Inside.—There was considerable coke on the inby and outby exposures of the shelves at E 240, E 340, and E 440; the rib was soot covered from E 300 to the face; at E 304 the coal was blistered, and many other small blisters were noted between this point and the face. The roof was badly blistered at E 440. A deposit of coked dust was found on the shelves at E 479. There was some coked dust on the outby end of the shelf at E 550. At E 590 there

were large quantities of coked dust on the outby exposures of the posts and bolt ends; at E 640 there was coked dust on the outby side of the posts and bolt ends; at E 654 there was coked dust on the outby corner and coke bubbles on the inby corner of the third cut-through.

LENGTH OF FLAME.

The length of flame, as indicated by guncotton tufts suspended along the entry, was 552 feet from the face to E 192 (fig. 22). The branch flame on the air course extended for 225 feet outby the third cut-through.

About 4 to 5 inches of stone dust remained on the shelves of the barrier after the explosion. The flame did not seem to be much checked by the barrier.

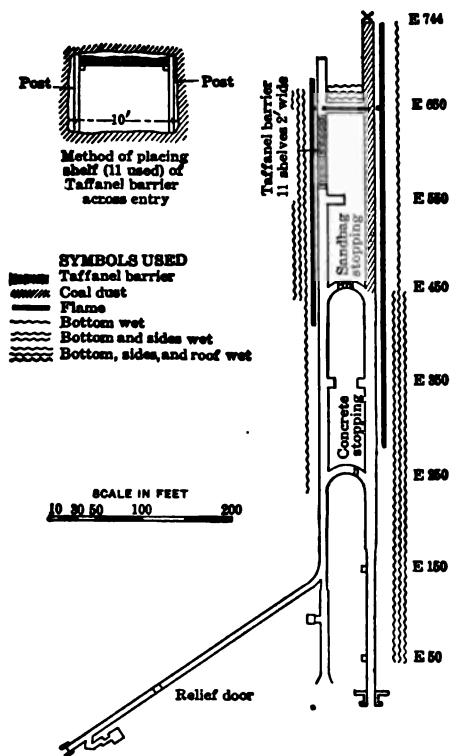


FIGURE 22.—Test 8, extent of flame and conditions of test.

SAMPLES ANALYZED.

A sample of the coal dust remaining on the shelves after the explosion was taken and analyzed, with results as follows:

Moisture	2.88
Volatile matter.....	32.43
Fixed carbon.....	51.88
Ash	12.81
	100.00
Sulphur	1.27

A comparison of the ratio of fixed carbon to volatile matter (38.47 : 61.53) with the ratio of fixed carbon to volatile matter of the original sample (38.47 : 61.53) shows that the dust remaining on the shelves

after the explosion was practically unaffected by the explosion. The increase in ash in the sample after the explosion was probably due to a mixing of ash from the consumed dust or road-dust material with the dust remaining on the shelves. Out of the 300 pounds put on the shelves for this test 87 pounds still remained after the explosion. As the quantity remaining on the shelves after this explosion was smaller than that remaining after the preceding

explosion, it would seem that the higher pressures developed caused more of the dust to be thrown in suspension.

A sample of dust that had settled out on the surface of a side support on which the shelves of the Taffanel barrier rest was taken and analyzed with results as follows:

Moisture.....	3. 21
Volatile matter.....	12. 24
Fixed carbon.....	11. 20
Ash.....	73. 35
	100. 00
Sulphur.....	. 64

The material of this sample had been in suspension immediately following the explosion, and the analysis indicates that the material was a mixture, probably of stone dust, coal dust, and soot.

TEST 9.

Date.—February 8, 1912.

Purpose.—In tests 6 and 8, pressure circuit breakers set to release at a pressure of 2 pounds were operated, seemingly by the pressure wave produced by the blown-out shot from the cannon. This test was made under the same conditions as those of the preceding test except that no coal-dust loading was used, so as to ascertain whether the same circuit breakers would be operated.

Point of origin of explosion.—Face of main entry, E 744.

Method of ignition.—Blown-out shot from bore hole of cannon.

Ignition charge.—Two and three-fourths pounds of FFF black blasting powder tamped with 4 inches of damp clay.

Coal dust.—No coal dust used in this test.

Length of flame.—The guncotton tufts at E 729 and E 704 were ignited by flame or flying pieces of burning powder.

INSTRUMENTS AND RECORDS.

INSTRUMENTS.

Location.—Pressure circuit breakers were located at stations E 50, E 150, E 250, E 350, E 450, E 550, and E 650.

Adjustment.—All the circuit breakers were adjusted to operate at a pressure of 2 pounds.

72322°—13—7

RECORDS.

The circuit breakers at E 450, E 550, and E 650 operated, the records being as follows:

Velocity of pressure wave, test 9.

Station.	Total time from ignition.	Time between stations.	Velocity between stations.
	<i>Seconds.</i>	<i>Seconds.</i>	<i>Feet per second.</i>
E 650.....	0.0683		
E 550.....	.1583	0.0900	1,111
E 450.....	.2493	.0910	1,099

It was now necessary to make further tests to ascertain the minimum pressure to which the circuit breakers could be adjusted and still not be operated by the pressure from a blown-out shot similar to the one fired in this test.

TEST 10.

Date.—February 9, 1912.

Purpose.—Determination of minimum pressure to which pressure circuit breakers could be adjusted and still not be operated by a blown-out shot from a cannon with the usual charge and stemming.

Point of origin of explosion.—Face of main entry, E 744.

Method of ignition.—Blown-out shot from cannon.

Ignition charge.—Two and three-fourths pounds of FFF black powder tamped with 4 inches of damp clay.

Coal dust.—No coal dust used in this test.

Inert zone.—The entry was sprinkled throughout its entire length in order that the road dust might be rendered inert.

Instruments.—Two pressure circuit breakers were installed at station E 650. No. 6 was adjusted to operate at a pressure of 3 pounds and No. 10 at a pressure of 4 pounds.

Results.—Both circuit breakers operated.

Length of flame.—Guncotton tufts at point E 704 were burned by flame or flying pieces of burning powder.

TEST 11.

Date.—February 9, 1912.

Purpose.—Determination of minimum pressure to which circuit breakers could be adjusted and still not be operated by a blown-out shot from a cannon with the usual charge and stemming.

Point of origin of explosion.—Face of main entry, E 744.

Method of ignition.—Blown-out shot from cannon.

Ignition charge.—Two and three-fourths pounds of FFF black blasting powder tamped with 4 inches of damp clay.

Coal dust.—None used.

Inert zone.—The entry was sprinkled for its entire length.

Instruments.—Two circuit breakers were installed at station E 650, one adjusted to operate at a pressure of 5 pounds and the other at a pressure of 6 pounds.

Results.—Neither of the pressure circuit breakers operated.

Conclusions.—In tests 9, 10, and 11 circuit breakers set to release at pressures of 4 pounds or less were operated by the pressure from the blown-out shot, whereas those set to operate at a pressure of 5 and 6 pounds did not operate. As a result of these tests it was decided in future tests in which the igniting shot was from the cannon with the usual charge and stemming to set the pressure circuit breakers to operate at a pressure of 5 pounds.

TEST 12.

Date.—February 10, 1912.

Purpose.—Study of the propagation of coal-dust explosions and the efficiency of Taffanel barriers.

Point of origin of explosion.—Face of main entry, E 744.

Method of ignition.—Blown-out shot from cannon.

Ignition charge.—Two and three-fourths pounds of FFF black blasting powder, tamped with 4 inches of damp shale dust.

COAL DUST.

Source.—The coal dust was ground from bituminous coal from the Pittsburgh bed at Willock, Pa.

Size.—95 per cent passed through a 100-mesh screen.

Analysis.—

Moisture.....	2. 13
Volatile matter.....	35. 29
Fixed carbon.....	55. 69
Ash.....	6. 89
	100. 00
Sulphur.....	1. 20

Zone.—From E 744 to E 34, or 710 feet.

Position.—E 742 to E 726 on a bench placed just below the level of the bore hole of the cannon; E 733 to E 34 on permanent side shelves on both sides of the entry.

Quantity.—Four hundred and eighty-two pounds. The first 290 feet was loaded at the rate of 1 pound per linear foot; the outer 420 feet was loaded at the rate of 0.457 pound per foot.

Inert zone.—The third cut-through and the air course were sprinkled; the floor of the main entry was also sprinkled to render the road dust inert.

TAFFANEL BARRIER.

Location.—In the air course, from 5 to 85 feet outby the third cut-through.

Number of shelves.—Thirteen.

Distance between shelves.—Six feet center to center.

Width of shelves.—Twenty inches.

Depth of shelves.—One inch.

Depth of stone dust.—Six and one-half inches.

Area of obstruction.—12.1 per cent.

Analysis of stone dust.—

Organic hydrogen.....	0.39
Organic carbon.....	7.59
Moisture (at 105° C.).....	3.94
Inorganic volatile matter (combined water, CO ₂ , etc.).....	5.42
Ash.....	82.66
	100.00

VENTILATION AND HUMIDITY.

At the time of the explosion about 5,000 cubic feet of air was moving through the last cut-through. A sample (No. 2252) of mine air, taken at the face of the main entry just prior to firing the shot, did not show any methane.

At the third cut-through the dry-bulb thermometer registered 35° F., and the wet-bulb thermometer 34° F., the humidity being 91 per cent

INSTRUMENTS AND RECORDS.

MANOMETERS.

Location.—Manometer A (suction) at station E 50; manometer B (pressure) at station E 50; manometer C (pressure) at station E 150.

Springs.—Manometers B and C were equipped with 200-pound springs.

PRESSURE CIRCUIT BREAKERS.

Location.—At stations E 50, E 150, E 250, E 350, E 450, E 550, and E 650.

Adjustment.—Set to operate at a pressure of 5 pounds.

FLAME CIRCUIT BREAKERS.

Type.—Detonator type with silver-carbide stick.

Location.—At stations E 50, E 150, E 250, E 350, E 450, E 550, and E 650.

RECORDS.

Pressure.—Too low to be recorded by manometers B and C. The record obtained on the suction manometer is reproduced in figure 23.

Velocity.—Pressure circuit breaker at station E 650 operated; flame circuit breakers at stations E 650, E 550, E 450, and E 350 operated. Tabulated results were as follows:

Velocity of flame, test 12.

Station.	Total time from ignition.	Time between stations.	Velocity between stations.
	<i>Seconds.</i>	<i>Seconds.</i>	<i>Feet per second.</i>
E 650.....	0.5972	0.6540	153
E 550.....	1.2512	.6210	161
E 450.....	1.8722	.4448	225
E 350.....	2.3170		

OUTSIDE MANIFESTATIONS.

The steel sheets of the explosion-door section were thrown out of place. No flame was noted at the main opening, but a cloud of smoke shot out 100 to 150 feet.

INSIDE OBSERVATIONS.

Little violence was noted. No shelves were thrown down. Only three bags of the stopping in the second cut-through were displaced. It was estimated that 4 to 5 inches of the stone dust remained on the shelves of the barrier.

LENGTH OF FLAME.

The flame extended from E 744 to E 279, or 465 feet (fig. 24). The length in the air course was 85 feet. It was noted that the flame did not extend beyond the Taffanel barrier.

AFTERDAMP.

The mine was filled with smoke and afterdamp; clearing it for the next explosion took 45 minutes.

SUCTION, POUNDS PER SQUARE INCH

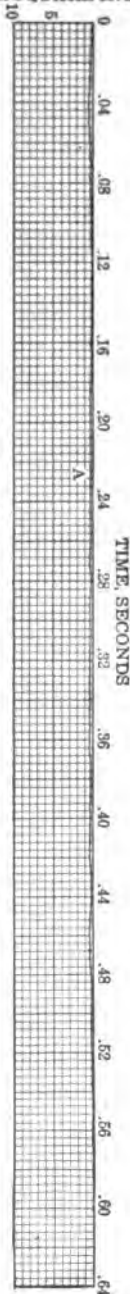


FIGURE 23.—Manometric curve, test 12, pressure below atmospheric.

TEST 13.

Date.—February 10, 1912.

Purpose.—To ascertain whether an explosion could be produced from dust that had been exposed to the previous explosion.

Point of origin of explosion.—Face of main entry, E 744.

Method of ignition.—Blown-out shot from 1½-inch pipe in coal.

Charge.—Two pounds of FFF black blasting powder, tamped with 4 inches of damp shale dust.

COAL DUST.

Source.—The dust was ground from bituminous coal from the Pittsburgh bed at Willock, Pa.

Size.—Ninety-five per cent passed through a 100-mesh screen.

Zone.—From E 744 to the main opening, 744 feet.

Position.—E 742 to E 726 on bench placed just below the level of the hole containing the igniting charge; E 733 to the main opening on side shelves or on the floor.

Quantity.—Twenty - five pounds of fresh coal dust was placed on the bench at the face. Throughout the remainder of the coal-dust zone

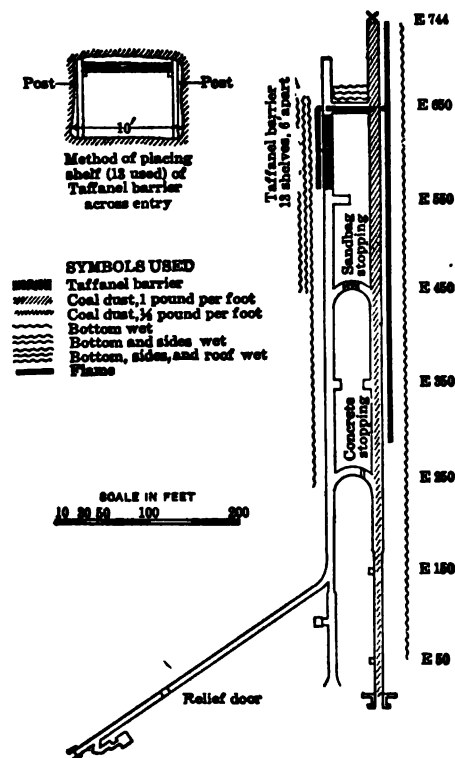


FIGURE 24.—Test 12, extent of flame and conditions of test.

the coal dust remaining after the preceding test was left as the load for this experiment.

TAFFANEL BARRIER.

Location.—In the air course, extending from 5 to 85 feet outby the third cut-through.

Number of shelves.—Thirteen.

Distance between shelves.—Six feet from center to center.

Width of shelves.—Twenty inches.

Depth of shelves.—One inch.

Depth of stone dust.—Four to five inches.

Analysis of stone dust.—

Organic hydrogen.....	0.39
Organic carbon.....	7.59
Moisture (at 105° C.).....	3.94
Inorganic volatile matter (combined water, CO ₂ , etc.).....	5.42
Ash.....	82.66
	100.00

OUTSIDE MANIFESTATIONS.

The steel sheets of the explosion-door section were disarranged. The smoke and coal dust shot out of the main opening for a distance or 100 to 150 feet.

INSIDE OBSERVATIONS.

Little violence was manifest. The curtain in the air course was blown down; a few of the bags in the cut-through were disarranged. The shelves of the Taffanel barrier were not disturbed.

LENGTH OF FLAME.

None of the guncotton tufts not affected by the preceding test was affected by this test, so that the length of the flame was manifestly not greater than in the preceding explosion. No recording instruments were used in this test.

TEST 14.

Date.—February 19, 1912.

Purpose.—To study the propagation of coal-dust explosions, and the efficiency of Taffanel barriers.

Point of origin of explosion.—Face of main entry, at E 744.

Method of ignition.—Blown-out shot from cannon.

Ignition charge.—Two and three-fourths pounds of FFF black blasting powder, tamped with 4 inches of damp shale dust.

COAL DUST.

Source.—The coal dust was ground from bituminous coal from the Pittsburgh bed at Willock, Pa.

Size.—Over 95 per cent passed through a 100-mesh screen.

Analysis.—

Moisture	3.30
Volatile matter.....	34.37
Fixed carbon.....	55.06
Ash.....	7.27
	100.00
Sulphur.....	1.25

Zone.—E 744 to E 34, or 710 feet.

Position.—E 742 to E 726 on bench placed just below the level of the bore hole of the cannon; E 733 to E 34 on permanent side shelves on both sides of the entry.

Quantity.—Six hundred and sixty pounds; loaded 0.93 pound per foot; equivalent to 0.248 ounce per cubic foot, or 248 grams per cubic meter.

Inert zone.—No watering was done for this test, but as there was a very large proportion of shale dust and clay in the road dust on the air course this was probably an inert zone.

TAFFANEL BARRIER.

Location.—In the air course, extending from 5 to 85 feet outby the third cut-through.

Number of shelves.—Thirteen.

Distance between shelves.—Six feet from center to center.

Width of shelves.—Twenty inches.

Depth of shelves.—One inch.

Depth of stone dust.—Six and one-half inches.

Analysis of stone dust—

Organic hydrogen.....	0.39
Organic carbon.....	7.59
Moisture (at 105° C.).....	3.94
Inorganic volatile matter (combined water, CO ₂ , etc.).....	5.42
Ash.....	82.66
	100.00

VENTILATION AND HUMIDITY.

At station E 550, where the cross section was 6½ by 9 feet, the velocity of the air was 104 feet per minute, the quantity being 6,084 cubic feet per minute. The humidity readings were as follows:

Humidity readings, test 14.

Station.	Wet bulb.	Dry bulb.	Humidity.
	° F.	° F.	Per cent.
Outside.....	45	49	74
E 250.....	43	45	86
E 550.....	47	48	93

INSTRUMENTS AND RECORDS.

MANOMETERS.

Location.—Manometer A (suction) at station E 50; manometer B (pressure) at station E 50; manometer C (pressure) at station E 150.

Springs.—Manometers B and C were equipped with 200-pound springs.

PRESSURE CIRCUIT BREAKERS.

Location.—At stations E 50, E 150, E 250, E 350, E 450, E 550, and E 650.

Adjustment.—Set to operate at a pressure of 5 pounds.

FLAME CIRCUIT BREAKERS.

Type.—Detonator type with silver-carbide stick.

Location.—At stations E 50, E 150, E 250, E 350, E 450, E 550, and E 650.

TIN-FOIL STRIPS.

In order to determine their suitability for use in flame circuit breakers, in this test four strips of tin foil of different thicknesses were placed on the shelves opposite stations E 50 to E 650, inclusive.

RECORDS.

Pressures.—The motors running the drums of the manometers failed to run; therefore only the maximum pressures were recorded. The records showed a pressure of 14.7 pounds at station E 150 and of 5.6 pounds at station E 50.

Velocity records.—The pressure circuit breakers at stations E 650, E 350, E 150, and E 50 operated; flame circuit breakers at stations E 650, E 450, E 350, and E 250 operated. The failure of the pressure and flame circuit breakers at station E 550 to operate prevented the obtaining of records of the circuit breakers at the stations outby this point. All tin-foil strips, except the two thickest at station E 50, were fused by the flame.

OUTSIDE MANIFESTATIONS.

A great cloud of dust swept out the main opening $1\frac{1}{2}$ seconds after the igniting shot had been fired. When the cloud had seemingly extended 50 to 75 feet from the opening, the flame shot through it, accompanied by a sharp crack, due probably to detonation of coal dust. The wave coming out of the gallery entry raised the steel sheets of the explosion-door section and piled them one upon another at the end of that section. The car dump standing at the top of the slate pile was blown over and landed down at the dam, one footboard having been broken. A barrel containing some ice was blown from a point 75 feet in front of the main entry to the fence near the slate dump. The "Danger" signboard on the air course was blown 100 feet distant. A piece of brattice cloth from the air-course curtain 75 feet inby the opening was blown outby to the air-course entrance.

INSIDE OBSERVATIONS.

A curtain packed in the crevices above the door frame inby the shaft was sucked inby 15 feet from its original position. An electric-light wire extending from the end of the concrete section in the air course to the first crosscut was blown outby so that it stretched from the end of the concrete to the intersection of the gallery and the air course.

Most of the dust on the Taffanel shelves was blown off. The first shelf outby the cut-through was blown down, and the second was swung around until it was parallel with the side of the entry.

In the main entry the mine track at E 530 seemed to have been raised slightly, as the ballast was very loose. This was possibly caused by the suction following the explosion wave.

The bags of the top 2 feet of the sandbag stopping in the second cut-through were blown out. Some bags were thrown toward the entry, but many more were thrown toward the air course. Many of these bags were upset and the sand spilled.

At E 390 the east end of one of the ties was bent forward 45° . This tie was broken where the rail was spiked to it. The rail had probably been slightly fractured at this point preceding the explosion. The post holding the shelves at E 294 was pushed into the groove in the rib $1\frac{1}{2}$ inches. The floor from

E 220 to E 160 was raised several inches, the ballast between the ties being rather loose.

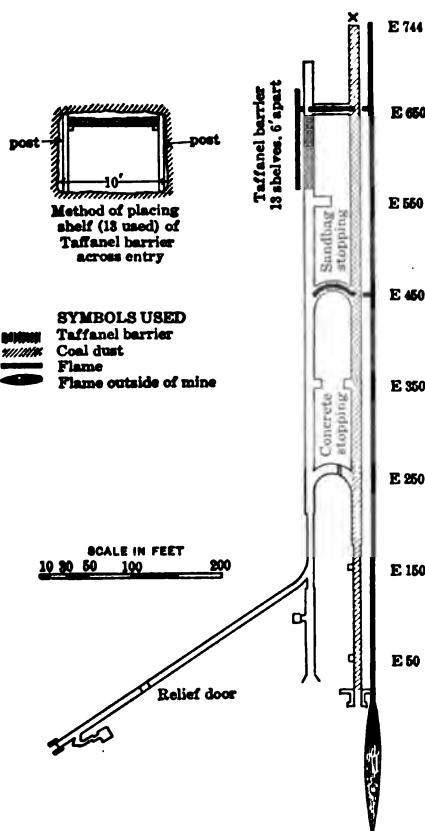


FIGURE 25.—Test 14, extent of flame and conditions of test.

LENGTH OF FLAME.

The flame extended through the full length of the entry (fig. 25). In the cut-through air-course branch it extended 130 feet, but did not pass the Taffanel barrier.

COAL DUST AND COKED DUST.

A thin layer of coal dust and soot was observed in the air course as far out as 36 feet inby the first cut-through. The whitewashed ribs inby this point were very much blackened and coated with dust and soot.

The first two posts of the Taffanel barrier on the east side outby the third cut-through had large crusts of coked dust on the inby side.

The coal was blistered slightly on the north side of the last cut-through.

There was a heavy crust of coked dust on the outby side of the first post on the east side of the entry outby the third cut-through.

A heavy crust of coked dust was found on the outby side of a post at E 658 on the east side of the entry.

A heavy coating of soot covered the shelves inby the third cut-through.

Scattered patches of coked dust were noted on the outby exposures of the coal and posts from E 650 to E 550.

There were crusts of coked dust on the outby sides of posts at E 550 west and E 540 east. There was some coked dust on the outby exposures of the ribs and posts from E 530 to E 440. There was a small deposit of coked dust on the outby side of the shelves at E 350. Slight quantities of coked dust were noted on the outby ends of the shelves at E 250.

ANALYSIS OF AIR SAMPLE.

A sample (No. 2251) of mine air was taken in the last cut-through as soon as it was possible for persons to enter without helmets. The sample was analyzed with results as follows:

CO ₂	0.08
O ₂	20.85
CO.....	.00
CH ₄	.03
N.....	79.04

It is to be noted that there was little black damp and no carbon monoxide in the air at the point where this sample was taken, yet the odor had been very irritating.

TEST 15.

Date.—February 24, 1912.

Purpose.—Public exhibition of the explosibility of coal dust and influence of a Taffanel barrier on the advance of the flame from an explosion.

Point of origin of the explosion.—Face of the main entry, E 747.

Method of ignition.—Blown-out shot from a 1½-inch pipe in the coal.

Charge.—Three pounds of FFF black blasting powder, tamped with 5 inches of damp shale dust.

COAL DUST.

Source.—Coal dust was ground from bituminous coal from the Pittsburgh bed; some of which was obtained from the Oak mine, near Pittsburgh, and the remainder from the mine at Willock, Pa.

Size.—Over 95 per cent passed through a 100-mesh sieve.

Analysis.—An analysis (No. 13389) of the coal dust from coal from Oak mine was made with the following results:

Moisture.....	2.43
Volatile matter.....	36.73
Fixed carbon.....	56.16
Ash.....	4.68
	100.00
Sulphur.....	.99

Zone.—In entry, E 747 to E 34, or 713 feet; in cut-through 50 feet; in air course, A 654 to A 427, or 227 feet, and A 347 to A 290, or 57 feet; total, 1,047 feet.

Position.—E 747 to E 731 on bench just below the level of the hole containing igniting shot; E 733 to E 34 on permanent side shelves, also on the floor; in cut-through and air course, on floor, and on three transverse "book" shelves.

Quantity.—Two thousand two hundred and thirty-two pounds, or an average of 2.13 pounds per linear foot; equivalent to 0.568 ounce per cubic foot or 568 grams per cubic meter.

Inert zones.—Air course, A 290 to opening; gallery slant.

TAFFANEL BARRIER.

Location.—A 427 to A 347.

Length.—Eighty feet.

Number of shelves.—Thirteen.

Distance between shelves.—Six to seven feet from center to center.

Width of shelves.—Twenty inches.

Thickness of shelves.—One inch.

Depth of stone dust.—Six and one-half inches.

Obstruction.—10.4 per cent of total cross section.

Analysis of stone dust.—

Organic hydrogen.....	0.39
Organic carbon.....	7.59
Moisture (at 105° C.).....	3.94
Inorganic volatile matter (combined water, CO ₂ , etc.).....	5.42
Ash.....	82.66
	100.00

VENTILATION AND HUMIDITY READINGS.

Ventilation.—At station E 540, where the cross section was 6 by 9 feet, the velocity of the air was 101 feet per minute, and the volume 5,445 cubic feet per minute.

Humidity readings, test 15.

Station.	Dry bulb.	Wet bulb.	Humidity.
	<i>° F.</i>	<i>° F.</i>	<i>Per cent.</i>
E 50.....	44½	43	89
E 250.....	45	43½	89
E 550.....	44	42	85
Air course.....	38	36	83

ANALYSIS OF SAMPLE OF AIR.

A sample of air was taken at face of entry just before firing the igniting shot. An analysis was made with the following results:

CO ₂	0.04
O ₂	20.95
CO.....	.00
CH ₄	Trace.
N.....	79.01
	100.00

INSTRUMENTS AND RECORDS.

MANOMETERS.

Location.—B. C. D. manometer A (suction) at station E 50; B. C. D. manometer B (pressure) at station E 50; B. C. D. manometer C (pressure) at station E 150.

Springs.—Manometers B and C were equipped with springs having a rating of 200 pounds.

PRESSURE CIRCUIT BREAKERS.

Location.—Pressure circuit breakers were installed in all stations from E 50 to E 650, inclusive.

Adjustment.—All pressure circuit breakers were adjusted so as to be operated by a pressure of 3 pounds.

FLAME CIRCUIT BREAKERS.

Type.—Tin foil. (See p. 63.)

Location.—At all stations from E 50 to E 650, inclusive.

SAMPLING BOTTLES.

Two types of sampling bottles were used in this test. The B. C. D. automatic sampling bottle was installed at station E 150, its circuit maker being installed at station E 250. A special sampling bottle was installed in a groove in the rib at E 374.

GUNCOTTON TUFTS.

Guncotton tufts were hung on wires from pegs in the roof at intervals of 25 feet throughout the length of the main entry and the air course.

RECORDS.

Manometers.—Records were obtained on all three manometer cylinders, which are reproduced in figures 26 and 27.

Pressure circuit breakers and flame circuit breakers.—All circuit breakers operated, but on account of the failure of the automatic commutators, records from only four stations were obtained from the pressure circuit breakers and from five stations from the flame circuit breakers. (See figs. 9, 10, and 11.)

Velocity of propagation of explosion, test 15.

Station.	Pressure wave.			Flame.		
	Total time from ignition.	Time between stations.	Velocity between stations.	Total time from ignition.	Time between stations.	Velocity between stations.
	<i>Seconds.</i>	<i>Seconds.</i>	<i>Feet per second.</i>	<i>Seconds.</i>	<i>Seconds.</i>	<i>Feet per second.</i>
E 650.....	0.0823	^a 0.0878	^a 1,139	2.174	0.245	408
E 550.....	.1701	2.085		2.419	.074	1,351
E 450.....	2.265	.102	990	2.493	.044	2,273
E 350.....	2.367			2.537	.047	2,128
E 250.....				2.584		
E 150.....	2.490	^b .125	^b 800			
E 50.....	2.605					

^a Due to shock from shot.

^b B. C. D. manometer records.

Sampling bottles.—The pipe of the B. C. D. automatic gas-sampling device at E 150, which projected from the plate to the center of the entry, was broken off at the plate and blown to some other place not known. An automatic gas-sampling device had been set in a recess in the rib at E 370; when the shelves at this point were thrown down, the pipe connection between the bottle and the valve was broken off, and this destroyed the automatic sampling device.

OUTSIDE MANIFESTATIONS.

The first indication of the explosion was the projection of a huge cloud of dust from the opening (Pl. I). When the advancing front of the cloud had reached a point about 100 feet from the opening a spear of flame (fig. 28) from the opening shot into the cloud and disappeared.

There was no manifestation at the other openings except the displacing of the steel sheets on the explosion-door section.

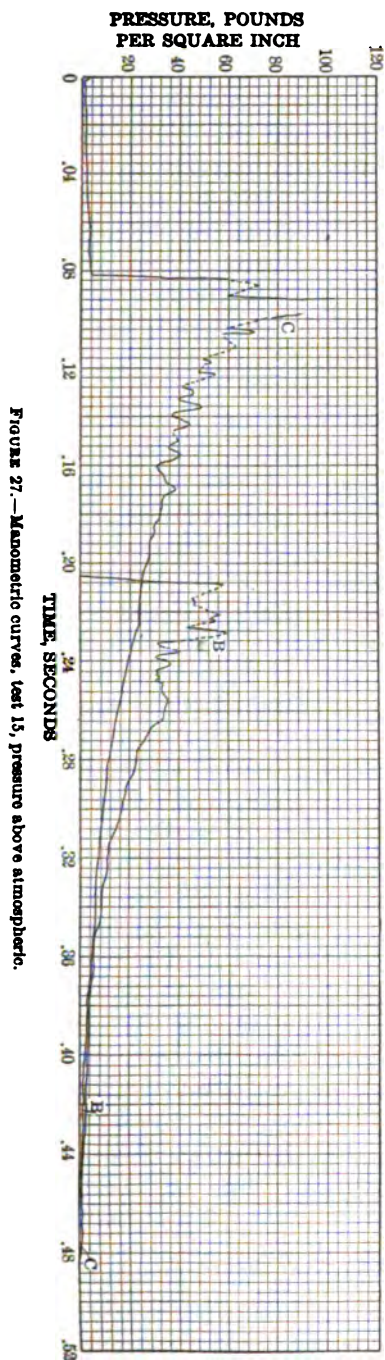


FIGURE 27.—Manometric curves, test 15, pressure above atmospheric.

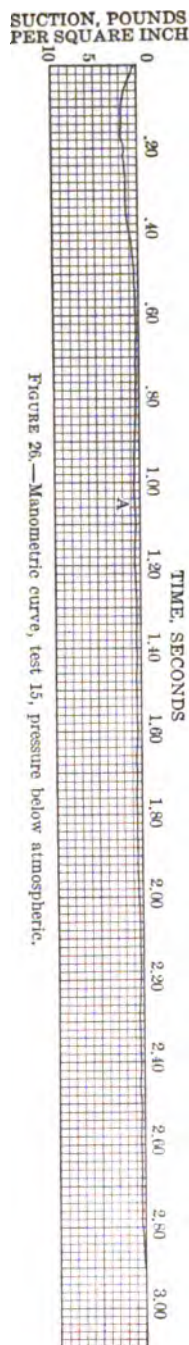


FIGURE 26.—Manometric curve, test 15, pressure below atmospheric.

An empty mine car that had stood on the track 33 feet in front of the opening was blown over the dump and landed in the valley beyond at a point 220 feet distant from its original position (Pl. V, B).

The gates at the end of the buttress wall at the main opening were broken and all but the upper portion of the west gate was demolished. The post from which the east gate had been hung, notwithstanding that it was recessed at the end of the buttress wall, was torn from position and carried 33 feet from the opening. All of the lights in the south and west side windows of the pump house were broken outward.

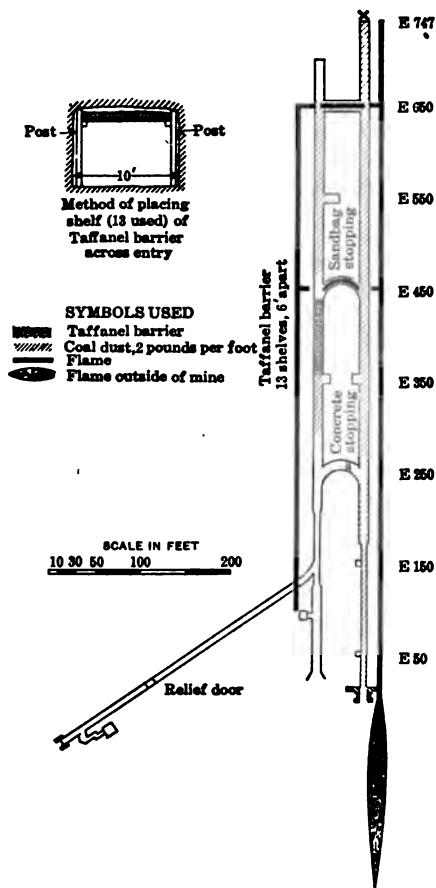


FIG. 28.—Test 15, extent of flame and conditions of test.

INSIDE INDICATIONS OF VIOLENCE.

In the concrete-lined portion of the main entry the same effect was observed after this explosion as had been noted after previous violent explosions, in that the arch of the section had been lifted, the consequent straightening of the vertical reinforcing rods having forced off pieces of concrete lying between these rods and the face of the concrete at the springing line. With one or two exceptions the effect was less than after the explosion of October 30, probably because the concrete had a much more solid backing over the arch, where a large quantity of cement grouting had been run in from the surface through holes. A piece of concrete 30 by 18 inches, with a thickness of from 2 to 3

inches, was forced off at the springing line just over the door at E 50; this piece of concrete was hanging from a $\frac{1}{4}$ -inch round rod in front of the door after the explosion.

At E 61 a $\frac{3}{4}$ -inch square reinforcing rod was buckled by the settling back of the arch, the extent of buckling indicating a raise of at least 3 inches.

At E 77 a crack $1\frac{1}{2}$ inches deep had opened between the original arch and the repair arch put inside the original one after the explosion

of October 30. Many of the small cracks caused by the explosion of October 30 were enlarged by this explosion.

The track from E 150 to E 170 seemed to have been raised by the suction following the explosion wave. The ballast was loose and springy throughout this length.

The shelving at the end of the concrete section on the west side of the entry was disarranged by the explosion. The first shelf at the top was broken 2 feet outby the post; the third shelf was broken 5 feet outby the post; and the fourth and fifth shelves were broken loose from the supporting posts at their outby end.

On the east side the third and fifth shelves were unfastened from their supports in the concrete.

Three shelving sets on the east side of the entry extending across the first cut-through were thrown down and moved out of place. The pressure wave seemed to have swept around the inby corner of the cut-through and carried the shelves that extended across the open space against the north rib of the cut-through. When the resulting pressure in the blind cut-through was relieved, a wave swung the shelves of the next set across the main entry and carried the supporting post across to the west rib.

Previous to the test a $\frac{1}{2}$ -inch square iron clamp, the ends of which were embedded in concrete in the rib, had held the post inby the cut-through in the groove in the rib. When this post was carried out of the groove the $\frac{1}{2}$ -inch bar was broken off where it passed around the post.

The posts and shelves at a dead end opening on the east side of the entry at E 350 were disarranged in like manner to those at E 250. The post on the inby corner was torn out, its clamps broken, and the shelves and post carried across the entry and outby 15 feet.

The sandbag stopping in the second cut-through was partly blown down, the bags of the upper 2 feet of the stopping having been carried in about equal proportion toward the entry and the air course. A cap piece, supported by braces to strengthen the sandbag stopping on the air-course side, was thrown down.

A straw dummy that had been set up at E 650 was torn apart and part of it thrown to the air-course end of third cut-through and another part to E 550.

The igniting charge at the face of the entry had cratered the hole 30 inches back from the front. The front of the hole was 4 feet wide by $4\frac{1}{2}$ feet high.

Some of the temporary track at the face was torn up by the explosion.

The "book" shelves that had been placed at intervals in the third cut-through and the air course were torn to pieces. Only the third, fourth, and seventh posts on the east side of the stone-dust

barrier remained standing after the explosion; all the rest of barrier was blown down and the shelves destroyed. The posts and side supports were scattered along the entry for 150 feet outby.

One of the track ties at A 330 was broken at the middle and the east end moved outby. Ties at A 290 and A 281 were also broken in like manner. These breaks were probably due to blows received from the flying posts of the stone-dust barrier.

COAL DUST AND COKED DUST.

No coke or coked dust was noted in the entry from the opening to E 550. Practically all of the coal dust had been swept from the shelves throughout this length, so that the blast was probably too great to allow deposition of coked dust. From E 550 to the face there were increasingly large deposits of soot on the shelves. There was a deposit of coked dust on the outby side of a post at E 648; also heavy deposits on the inby corner of the third cut-through and the entry. At E 656 there were smaller deposits on the inby side of a post on the east side of the entry, and on the outby side of a post on the west side of the entry.

The coal in the entry near the cut-through and through the cut-through had many new blisters. The east rib of the air course opposite the third cut-through and for a short distance outby this point was badly blistered.

In the air course there were still large quantities of coal dust remaining on the floor from A 650 to A 450 but in decreasing amount as A 450 was approached. The coal dust overlaid the shale dust from A 450 to A 350. There was considerable coal dust still remaining on the floor for 75 feet outby this point but very little beyond.

A sample of the coked dust from the inby corner of the main entry and the third cut-through was analyzed with the following results:

Moisture.....	1.10
Volatile matter.....	15.40
Fixed carbon.....	43.19
Ash.....	40.31
	100.00
Sulphur	1.54

The ratio of volatile matter to fixed carbon shows a large decrease in the percentage of volatile matter over that in the original coal dust.

EFFECT OF BARRIER.

It is not known precisely how far the flame extended beyond the Taffanel barrier. A guncotton tuft 30 feet inby the mouth of the air course was not burned, whereas at 55 feet inby the air-course

mouth neither the tuft nor the wire that had held it could be found. The flame had burned the guncotton at a point 80 feet from the mouth, so that the flame must have extended past A 80 but not beyond A 30. Although the barrier did not stop the flame immediately, it had such an effect that the flame extended a much shorter distance than previous experiments indicated it would have gone.

ANALYSES OF GAS SAMPLES TAKEN AFTER THE EXPLOSION.

Two samples of air were taken by a helmet party that entered the mine at the main opening about 15 minutes after the explosion and while the smoke was still issuing.

The analysis of a sample (No. 2249) taken 150 feet in by the main opening showed the following results:

CO ₂	1. 59
O ₂	18. 38
CO.....	1. 05
CH ₄	. 42
N.....	78. 56
	100. 00

The analysis of a sample (No. 2250) taken 200 feet in by the main opening showed results as follows:

CO ₂	1. 47
O ₂	18. 30
CO.....	1. 08
CH ₄	. 35
N.....	78. 64
H ₂	. 16
	100. 00

The results of the two analyses are very similar. The oxygen content is high and would be capable of supporting life if no carbon monoxide were present, but the carbon monoxide content is so great that a person not protected by a helmet would retain consciousness only a few minutes in such an atmosphere.

PUBLICATIONS ON MINE ACCIDENTS AND TESTS OF EXPLOSIVES.

The following Bureau of Mines publications may be obtained free by applying to the Director, Bureau of Mines, Washington, D. C.:

BULLETIN 10. The use of permissible explosives, by J. J. Rutledge and Clarence Hall. 1912. 34 pp., 5 pls.

BULLETIN 15. Investigations of explosives used in coal mines, by Clarence Hall, W. O. Snelling, and S. P. Howell, with a chapter on the natural gas used at Pittsburgh, by G. A. Burrell, and an introduction by C. E. Munroe. 1911. 197 pp., 7 pls.

BULLETIN 17. A primer on explosives for coal miners, by C. E. Munroe and Clarence Hall. 61 pp., 10 pls. Reprint of United States Geological Survey Bulletin 423.

BULLETIN 20. The explosibility of coal dust, by G. S. Rice, with chapters by J. C. W. Frazer, Axel Larsen, Frank Haas, and Carl Scholz. 204 pp., 14 pls. Reprint of United States Geological Survey Bulletin 425.

BULLETIN 44. First national mine-safety demonstration, by H. M. Wilson and A. H. Fay, with a chapter on the explosion at the experimental mine, by G. S. Rice. 1912. 75 pp., 7 pls.

BULLETIN 46. An investigation of explosion-proof mine motors, by H. H. Clark. 1912. 44 pp., 6 pls.

BULLETIN 48. The selection of explosives used in engineering and mining operations, by Clarence Hall and S. P. Howell. 1913. 50 pp., 3 pls.

BULLETIN 52. Ignition of mine gases by the filaments of incandescent electric lamps, by H. H. Clark and L. C. Halsey. 1913. 31 pp., 6 pls.

TECHNICAL PAPER 4. The electrical section of the Bureau of Mines; its purpose and equipment, by H. H. Clark. 1911. 12 pp.

TECHNICAL PAPER 6. The rate of burning of fuse as influenced by temperature and pressure, by W. O. Snelling and W. C. Cope. 1912. 28 pp.

TECHNICAL PAPER 7. Investigations of fuse and miners' squibs, by Clarence Hall and S. P. Howell. 1912. 19 pp.

TECHNICAL PAPER 11. The use of mice and birds for detecting carbon monoxide after mine fires and explosions, by G. A. Burrell. 1912. 15 pp.

TECHNICAL PAPER 12. The behavior of nitroglycerin when heated, by W. O. Snelling and C. G. Storm. 1912. 14 pp., 1 pl.

TECHNICAL PAPER 13. Gas analysis as an aid in fighting mine fires, by G. A. Burrell and F. M. Seibert. 1912. 16 pp.

TECHNICAL PAPER 17. The effect of stemming on the efficiency of explosives, by W. O. Snelling and Clarence Hall. 1912. 20 pp.

TECHNICAL PAPER 18. Magazines and thaw houses for explosives, by Clarence Hall and S. P. Howell. 1912. 34 pp., 1 pl.

TECHNICAL PAPER 19. The factor of safety in mine electrical installations, by H. H. Clark. 1912. 14 pp.

TECHNICAL PAPER 21. The prevention of mine explosions; report and recommendations, by Victor Wetteyne, Carl Meissner, and Arthur Desborough. 12 pp. Reprint of United States Geological Survey Bulletin 369.

TECHNICAL PAPER 23. Ignition of mine gas by miniature electric lamps, by H. H. Clark. 1912. 5 pp.

TECHNICAL PAPER 24. Mine fires; a preliminary study, by G. S. Rice. 1912. 51 pp.

TECHNICAL PAPER 28. Ignition of mine gases by standard incandescent lamps, by H. H. Clark. 1912. 6 pp.

TECHNICAL PAPER 29. Training with mine-rescue breathing apparatus, by J. W. Paul. 1912. 16 pp.

TECHNICAL PAPER 40. Metal-mine accidents in the United States during the calendar year 1911, compiled by A. H. Fay. 1913. 54 pp.

TECHNICAL PAPER 46. Quarry accidents in the United States during the calendar year 1911, compiled by A. H. Fay. 1913. 34 pp.

TECHNICAL PAPER 47. Portable electric mine lamps, by H. H. Clark. 1913. 12 pp.

TECHNICAL PAPER 48. Coal-mine accidents in the United States, 1896-1912, with monthly statistics for 1912, compiled by F. W. Horton. 1913. 72 pp. 10 figs.

MINERS' CIRCULAR 3. Coal-dust explosions, by G. S. Rice. 1911. 22 pp.

MINERS' CIRCULAR 4. The use and care of mine-rescue breathing apparatus, by J. W. Paul. 1911. 24 pp.

MINERS' CIRCULAR 5. Electrical accidents in mines; their causes and prevention, by H. H. Clark, W. D. Roberts, L. C. Halsey, and H. F. Randolph. 1911. 10 pp., 3 pls.

MINERS' CIRCULAR 6. Permissible explosives tested prior to January 1, 1912, and precautions to be taken in their use, by Clarence Hall. 1912. 20 pp.

MINERS' CIRCULAR 9. Accidents from falls of roof and coal, by G. S. Rice. 1912. 16 pp.

MINERS' CIRCULAR 10. Mine fires and how to fight them, by J. W. Paul. 1912. 14 pp.

MINERS' CIRCULAR 11. Accidents from mine cars and locomotives, by L. M. Jones. 1912. 16 pp.



Bulletin 57

DEPARTMENT OF THE INTERIOR

BUREAU OF MINES

JOSEPH A. HOLMES, DIRECTOR

SAFETY AND EFFICIENCY

IN

MINE TUNNELING

BY

DAVID W. BRUNTON

AND

JOHN A. DAVIS



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II

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SAFETY AND EFFICIENCY IN MINE TUNNELING.

By **DAVID W. BRUNTON** and **JOHN A. DAVIS.**

INTRODUCTION.

PURPOSE OF REPORT.

During the past few years great progress has been made in the United States toward safer, more efficient, and more economical tunneling methods. This advance is partly due, no doubt, to the recent increase in the number of tunnels and adits driven for developing and draining mines and transporting ore. The Bureau of Mines during 1911 and 1912 made a special examination of this phase of mining operations, in connection with an investigation of mining methods and means for preventing accidents. The details especially studied were the provisions for the safety of employees, the kinds of equipment, the methods of driving, and the costs of construction. The results and conclusions obtained from that investigation are discussed and summarized in this bulletin.

Up-to-date information concerning tunneling methods is difficult to obtain. There are few books on the subject; and much of the material they contain, although interesting and of value historically, is now obsolete. The engineering periodicals, it is true, endeavor to keep abreast of the times, and several in nearly every issue present some article bearing upon tunnel work. But the very multiplicity of these articles prevents one from reading them all, and the foreman or superintendent in charge of a tunnel, or the mining engineer designing one, and especially the business man financing the project, has no time for a lengthy search after scattered articles in order to determine the present status of tunnel work. Then, too, knowledge of new methods travels slowly. Valuable inventions and improvements in tunneling as well as in all the other industries frequently remain in the notebook of the investigator, or as theses are buried in university libraries, or are published only in journals of small scientific societies. New tunneling methods and equipment that are proving safe, efficient, and economical may be totally unknown outside the district in which they originate. This paper, based on a special examination both in the field and in engineering literature, is intended to supply data concerning modern and recent tunneling practice in the United States, and to make suggestions that, it is hoped, will result in a saving to the mining industry of energy, capital, and life.

In most of the published accounts of tunnel work the writers do not attempt to criticize the methods they are describing. The articles usually present accurate descriptions of equipment and of various phases of working operations, with occasional figures showing the cost of the work; rarely do they include a discussion of the means for preserving the health and life of the employees or data bearing on the choice and efficiency of equipment, or an analysis of methods and costs. As a result, the reader who endeavors to draw conclusions is dependent wholly on his own resources. In this bulletin, on the contrary, the making of such analyses will be a primary consideration, but as the purpose of the bulletin is to give an impartial disinterested report, the criticisms made are intended to be constructive rather than destructive. For that reason emphasis is placed on safe, efficient, and economical methods and on good points of equipment, whereas bad practice and obsolete machinery are ignored, except, perhaps, as examples of what was inadvisable or as having some bearing historically. Thus the authors hope to set forth a guide for future work rather than an unilluminated record of past or present achievement.

SCOPE OF REPORT.

This bulletin is confined chiefly to a discussion of tunnels and adits for mining purposes, such as drainage, transportation, or development, but it also discusses those used to carry water for power, irrigation, or domestic use, in which the essential features are practically identical with mine tunnels.

Most tunnels of the sort discussed are driven through rocks at least fairly hard in contrast to ordinary soil, quicksand, and other heavy material of a treacherous nature, and practically none is driven through recent river-bed deposits. Therefore descriptions of the special methods and equipment for tunnel work in such materials are omitted. A distinction is made between tunnels or adits for which the excavation is wholly or in a large part in material containing no ore and those that follow a vein. As far as possible the discussion is limited to the former, because the methods employed in driving along a vein are usually more akin to the distinctive operations for removing ore and are therefore not so apt to be good examples of tunnel practice.

It has been suggested by prominent authorities that the word "tunnel" be restricted to the designation of such nearly horizontal passageways as extend completely through a mountain or hill from daylight to daylight, and the words "adit" and "drift" be used only for nearly horizontal galleries that enter from the surface and serve to drain a mine or furnish an exit from the workings but do not continue entirely through the hill. Such definition is eminently desirable from strict technical consideration, and would contribute to pre-

cision of usage, but, although the suggestion was made over 30 years ago and has been repeated several times since, such usage is not widely established. The American practice of referring to any horizontal gallery as a tunnel, without regard to whether it extends completely through a hill, is so firmly fixed in mining literature and among mining men in this country, even being embodied in the United States mining laws, that the use of a more precise definition has been thought scarcely justifiable in this report.

ACKNOWLEDGMENTS.

In the preparation of this bulletin the writers are deeply indebted to the officials of the New York Board of Water Supply, of the Los Angeles Aqueduct, and of the United States Reclamation Service, and to the officers, managers, superintendents, and foremen at the different tunnels for favors granted, for information supplied, often at no little inconvenience, and above all, for a hearty cooperation that has been an unfailing source of inspiration. Many thanks, also, are due the manufacturers of equipment and materials used in tunnel work for their promptness and courtesy in furnishing catalogues, data of tests, and similar material, and in supplying photographs, blue prints, and cuts. Acknowledgment is made of many valuable suggestions obtained from articles in engineering periodicals and from books on tunneling and related subjects.

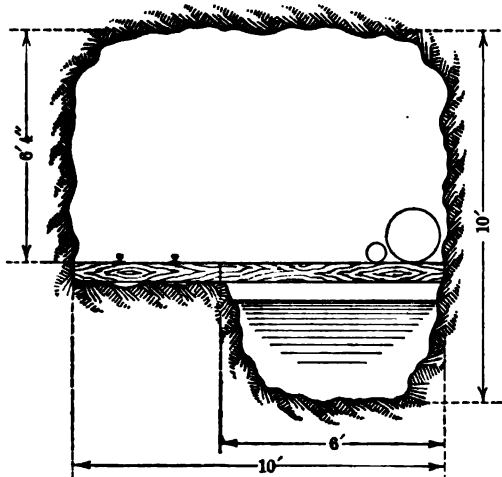


FIGURE 1.—Cross section of Roosevelt Tunnel.

TUNNELS VISITED.

The tables below briefly summarized the chief features of the tunnels and adits actually visited in the special field work upon which this report is based. Complete information was obtained wherever possible concerning surface and underground equipment, provisions for safety of the men, use of explosives, and methods employed in driving with regard to efficiency, cost, and other data bearing on construction.

Lack of space prevents a full presentation, but the tabulation conveys an adequate idea of the location, purpose, and magnitude of the different tunnels.

TABLE 1.—Description of tunnels visited.

Name of tunnel.	Location.	Operator.	Purpose.	Shape of cross section.	Size.	Length.	Character of rock penetrated.	Remarks.
Burleigh.....	Silver Plume, Colo.	Dives-Pelican & Seven-Thirty Mining Co.	Mine drainage and development.	Rectangular	6 feet wide, 7 feet high.	<i>Feet.</i> 3,000	Granite and gneiss.	Started by hand in fall of 1898; Burleigh drills introduced early in 1900. This was the first use of power drills for driving a mine adit in this country.
Carter.....	Ohio City, Colo.	Carter Mining Co.	Mine drainage and transportation.	Rectangular with arched roof.	5½ feet wide, 7½ feet high.	7,600	Gneiss, granite, and porphyry.	Started July, 1897; on Nov. 1, 1911, driven 6,450 feet. Part of interesting time spent in drifting laterals; 3 years shut down entirely, and 5 years only three men at work.
Central.....	Idaho Springs, Colo.	Big Five Tunnel, Ore Reduction & Transportation Co.	do.....	Rectangular	2,600 feet of tunnel, 12 feet wide, 8 feet high; remainder 5 feet wide, 7 feet high.	9,000	Idaho Springs gneiss.	
Gold Links.....	Ohio City, Colo.	Colorado Smelting & Mining Co.	do.....	Rectangular with arched roof.	6 feet wide, 8 feet high.	3,900	Gneiss, intruded granite, and porphyry.	Started May, 1906. Driven intermittently until completed, December, 1912.
Gunnison.....	Montrose, Colo.	U. S. Reclamation Service.	Irrigation.....	Horseshoe.....	10 feet wide at bottom; 10 feet 6 inches at spring line; 10 feet high at spring line; 12½ feet high at center of arch.	30,645	Chiefly metamorphosed granite with some water-bearing clay and gravel, some hard black shale, and a zone of faulted and broken material.	Began Jan. 11, 1906; headings holed through July 6, 1909; trimmed to full size February, 1910.
Laramie-Poudre.	Home, Colo.	Laramie-Poudre Reservoir & Irrigation Co.	do.....	Rectangular	9½ feet wide, 7½ feet high.	11,308	Close-grained red and gray granite.	Started Dec. 25, 1909; completed July 20, 1911.
Lausanne.....	Manch Chumk, Pa.	Laugh Coal & Navigation Co.	Mine drainage.....	Arched roof...	12 feet wide, 4 feet high at spring line; 8 feet high at center of arch.	20,000	Manch Chumk red shale, Pottsville conglomerate, slate, and anthracite coal.	Started in the fall of 1901 and driven intermittently up to 1907, suspended from summer of 1907 to summer of 1910; tunnel driven up to Dec. 1, 1911, 6,385 feet.
Locust.....	Idaho Springs, Colo.	Locust Tunnel & Mines Co.	Mine development and transportation.	Square.....	8 by 8 feet.....	12,000	Hard granite.....	

Marshall - Russell.	Empire, Colo..	Marshall - Russell Gold Mining, Milling & Tunneling Co., Board of water commissioners, City of Santa Barbara.	Mine drainage, development, mining, and transportation. Water supply.	Rectangular.	8 feet wide, 9 feet high.	Granite and gneiss.	Started October, 1901; 1,000 feet driven intermittently up to 1907; since then regularly, 6,400 feet completed, 1912.
Mission.	Santa Barbara, Cal.			Trapezoid.	4½ feet widest at top; 6 feet wide at base; 7 feet high.	Shale, slate, and hard sandstone.	
Newhouse.	Idaho Springs, Colo.	Idaho Springs Drainage & Tunneling Co.	Drainage and transportation.	Square.	8 by 8 feet.	Idaho Springs gneiss.	Started by hand September, 1892. Machines installed January, 1894. Operations suspended September, 1897; resumed Oct. 1, 1899; suspended Mar. 31, 1902; resumed June 20, 1904; suspended Apr. 30, 1905; resumed Apr. 1, 1906; suspended Feb. 22, 1907; resumed Dec. 16, 1908; suspended Dec. 31, 1909; resumed Apr. 1, 1910; completed Nov. 17, 1910. Progress Nov. 1, 1911 (two headings), 8,923 feet; completed 1912.
Nisqually.	Alder, Wash.	Nisqually Contract Co.	Hydroelectric power for city of Tacoma.	Rectangular with arched roof.	9½ feet wide; 11 feet high at center walls 9 feet high.	Rhyolite.	Started July 26, 1888; suspended several times for 1 to 14 months. Still unfinished.
Ontario.	Park City, Utah.		Mine drainage.	Trapezoid.	5 feet widest at base; 4 feet wide at top; 6 feet high; above track; ditch 1½ feet deep, 6 feet wide, below tracks.	Porphyry quartzite, limestone, and granite.	
Rawley.	Bonanza, Colo.	Rawley Mining Co.	Mine drainage and development.	do.	8 feet widest at base; 7 feet wide at top; 7 feet high.	Andesite.	Started May 27, 1911. On Nov. 1 driven 1,040 feet, and on Dec. 1 2,335 feet. Completed October 1912.
Raymond.	Ohio City, Colo.	Raymond Consolidated Mines Co.	do.	Square.	9 by 9 feet.	Gneiss to 1,000 feet, then granite and schist with porphyry inclusions.	Started December, 1903; suspended March, 1908, to June, 1910. Length of tunnel driven Dec. 1, 1911, 2,300 feet. Completed 1912.
Roosevelt.	Cripple Creek, Colo.	Cripple Creek Drainage & Tunnel Co.	Mine drainage.	See figure 1.		Pikes Peak granite with some phonolite schist near end.	Started June, 1907; work progressed steadily until January, 1911, when discontinued at 15,743 feet. Resumed October, 1911, and advanced 375 feet (to 16,118 feet) on Dec. 1, when work still in progress.
Slwath.	Leadville, Colo.	Slwath Mining & Tunnel Co.	Development.	Rectangular.	6 feet wide by 7½ feet high.	Granite.	

b Projected.

a Approximate.

TABLE 1.—Description of tunnels visited—Continued.

Name of tunnel.	Location.	Operator.	Purpose.	Shape of cross section.	Size.	Length.	Character of rock penetrated.	Remarks.
Snake Creek.....	Heber, Utah..	Snake Creek Mining & Tunnel Co.	Mine drainage and development.	Rectangular with ditch on side.	9½ feet wide by 6½ feet high.	<i>Fet.</i> a 14,000	Diabase.....	Started May, 1910; suspended July, 1911, at 4,100 feet, because of bad ground near 3,000-foot station requiring reinforced concrete, which was completed about Jan. 1, 1912, and driving resumed.
Sulwell.....	Telluride, Colo.	Liberty Bell Gold Mining Co.	do.....	Square, with ditch at side.	7 by 7 feet.....	2,500	Conglomerate and andesite.	Started September, 1906; completed in 1906.
Strawberry.....	Strawberry Valley, Utah and Wasatch Counties, Utah..	U. S. Reclamation Service.	Irrigation.....	Straight bottom walls and arched roof.	8 feet wide by 9½ feet high.	19,100	Limestone and interbedded sandstones for 10,000 feet, then sandstone and interbedded shale.	Started 1906; driven intermittently with temporary equipment until beginning of 1909; since then steadily from west heading only. On Oct. 1, 1911, had driven 11,168 feet. Completed in 1912.
Utah Metals.....	Tooele, Utah..	Utah Metal Mining Co.	Transportation.....	Rectangular	10 feet wide, 8 feet high.	b 11,780	Quartzite.....	Started Nov. 27, 1909; suspended Apr. 13, 1910, to Dec. 9, 1909. Until Apr. 26, 1910, being enlarged from 4½ by 7 feet to 8 by 10 feet. Completed 8½ by 10 feet on Apr. 27, 1910. On Nov. 1, 1911, had driven 6,050 feet.
Yak.....	Leadville, Colo.	Yak Mining Milling & Tunnel Co.	Transportation and development.	Square.....	7 by 7 feet.....	23,800	Sandstone, limestone, shale, porphyry, and granite.	Started 1888. From 1888 to 1898 operated intermittently; from 1898 to completion in 1910 work practically continuous.

a Projected.

TUNNELS OF CATSKILL AQUEDUCT.

The Catskill Aqueduct (fig. 2), in Ulster, Orange, Putnam, and Westchester Counties, and the city of New York, N. Y., and operated

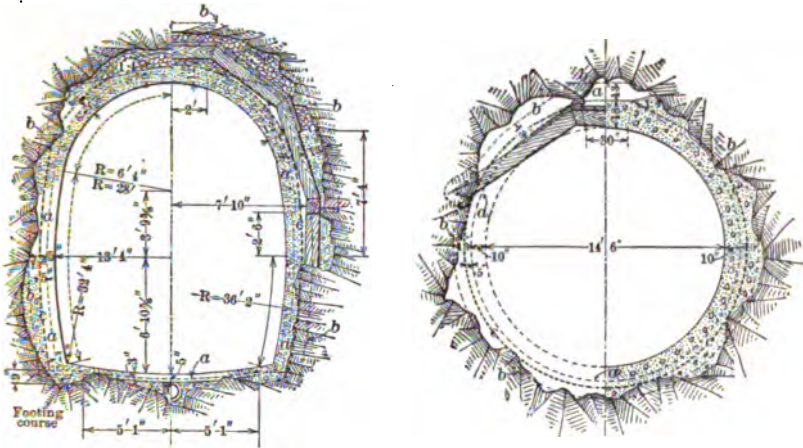


FIGURE 2.—Cross sections of typical tunnels in the Catskill Aqueduct. In the figure *a* is the line within which contractor was not permitted to leave projecting rock, *b* is the ideal average line of excavation, from which excavation, masonry, and packing quantities were computed, and *c* is the line of effective thickness of masonry, and trimming line, except for points on limited areas.

by the board of water supply, New York City, includes the following tunnels, of which those examined in the field are typical:

TABLE 2.—*Tunnels of Catskill Aqueduct.*

Name of tunnel.	Length.	Character of rock penetrated.	Started.	Completed.
	<i>Feet.</i>			
Peak.....	3,470	Hard rock.....	November, 1908....	November, 1909.
Rondout Siphon.....	23,608	Onondaga limestone, Binnewater sandstone, Hudson River shale, Esopus shale, High Falls shale, Shawangunk grit, Hamilton and Marcellus shale, Helderberg limestone.	March, 1909.....	May, 1911.
Bonticou.....	6,823	Hudson River shale, sandstone.	November, 1908....	February, 1911.
Walkkill Siphon.....	23,391	Hudson River shale.....	May, 1909.....	December, 1910.
Moodna Siphon.....	25,200	Hard sandstone, granite, and Hudson River shale.	February, 1909....	June, 1911.
Hudson Siphon.....	3,022	Granite.....	December, 1910....	January, 1912.
Breakneck.....	1,054	Granite, gneiss.....	September, 1910....	April, 1911.
Bull Hill.....	5,365	Granite.....	June, 1909.....	January, 1911.
Garrison.....	11,430	Hard gneiss.....	June, 1907.....	9,900 feet, December, 1911.
Hunters Brook.....	6,150	Hard rock, soft in places; schist.	September, 1909....	5,963 feet, December, 1911.
Turkey Mountain.....	1,400	Manhattan schist.....	October, 1909.....	December, 1910.
Croton Lake Siphon.....	2,639	Manhattan schist, Fordham gneiss.	July, 1910.....	January, 1912.
Croton.....	3,000	Manhattan schist.....	August, 1909.....	December, 1911.
Chadseyin.....	700	do.....	November, 1909....	September, 1910.
Millwood.....	4,750	Very hard rock, except 170 feet soft rock and earth; gneiss.	May, 1910.....	2,908 feet, December, 1911.

* Not visited in the field; data obtained from the New York office of the Board of Water Supply.

† Suspended November, 1910, to April, 1911.

TABLE 2.—*Tunnels of Catskill Aqueduct—Continued.*

Name of tunnel.	Length.	Character of rock penetrated.	Started.	Completed.
Series ^a	<i>Fet.</i> 5,230	Gneiss, hard rock, schist....	February, 1910....	3,558 feet, December, 1911.
Harlem R. R. ^a	1,100	do.....	June, 1910.....	January, 1911.
Reynolds Hill ^a	3,650	Schist.....	October, 1910.....	682 feet, December, 1911.
East View ^a	5,388	do.....	April, 1910.....	5,252 feet, December, 1911.
Elmsford ^a	2,375	Soft rock, schist.....	May, 1911.....	438 feet, December, 1911.
Yonkers Siphon.....	12,302	Yonkers gneiss.....	June, 1910.....	11,923 feet, December, 1911.
Van Cortland Siphon... City.....	1,809 b 18.11	do..... Fordham gneiss, Manhattan schist.	July, 1910..... December, 1911....	September, 1911.

^a Not visited in the field; data obtained from the New York office of the Board of Water Supply.

^b Miles.

TUNNELS OF LOS ANGELES AQUEDUCT.

The Los Angeles Aqueduct (fig. 3), in Inyo, Kern, and Los Angeles counties, Cal., and operated by the department of public works,

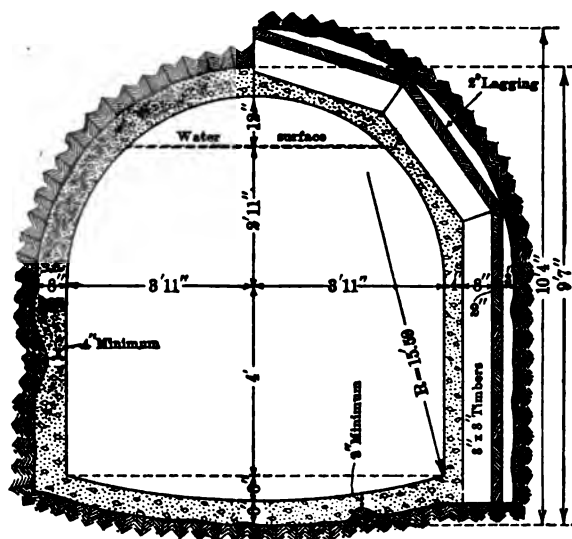


FIGURE 3.—Cross section of a typical tunnel, Los Angeles Aqueduct.

Los Angeles, for water supply, power, and irrigation, includes the following tunnels, which were examined in the field: ^a

^a This aqueduct includes a number of other tunnels that were not visited because they were either completed at the time of field examination or were being driven by hand drilling through soft material.

TABLE 3.—*Tunnels of Los Angeles Aqueduct visited.*

Name of tunnel.	Length.	Character of rock penetrated.	Started.	Completed.
Little Lake division:	<i>Feet.</i>			
Tunnel 1B.....	1,918	Medium hard granite....	South heading, June, 1909; north heading, July, 1909.	December, 1909.
Tunnel 2.....	1,739	Medium hard granite, very wet.	May, 1909.....	September, 1909.
Tunnel 2A.....	1,322	Medium hard granite....	do.....	September, 1909.
Tunnel 3.....	4,044	North heading, medium hard granite; south heading, granite of varying hardness with pockets of CO ₂ gas.	South heading, March, 1909; north heading, May, 1909.	July, 1911.
Tunnel 4.....	2,033	Medium hard to hard granite.	February, 1909.....	November, 1909.
Tunnel 5.....	1,178	Medium hard to very hard granite.	do.....	July, 1909.
Tunnel 6.....	411	Medium hard granite....	do.....	May, 1909.
Tunnel 7.....	3,596	Variable, soft and swelling in parts.	North heading, March, 1909; south heading, November, 1909.	July, 1911.
Tunnel 8.....	2,560	Medium hard to hard, swelling in parts.	North heading, November, 1909; south heading, December, 1909.	August, 1911.
Tunnel 9.....	3,506	Medium hard to hard...	November, 1909.....	February, 1911.
Tunnel 10.....	5,755	Medium hard granite....	South heading, December, 1909; north heading, January, 1910.	August, 1911.
Tunnel 10A.....	5,961	Medium hard to hard...	Both headings, March, 1910.	December, 1911.
Grape Vine division:				
Tunnel 12.....	4,900	Hard granite.....	July, 1909.....	May, 1911.
Tunnel 13.....	1,958	do.....	North heading, July, 1909; south heading, May, 1909.	April, 1910.
Tunnel 14.....	859	Hard rock.....	North heading, April, 1909; south heading, May, 1909.	February, 1910.
Tunnel 15.....	895	do.....	North heading, May, 1909; south heading, July, 1909.	December, 1909.
Tunnel 16.....	2,723	do.....	North heading, July, 1909; south heading, April, 1909.	February, 1910.
Tunnel 17.....	3,024	Hard granite.....	March, 1909.....	November, 1910.
Tunnel 17A.....	1,364	do.....	January, 1910.....	Oct. 31, 1910.
Tunnel 17A.....	5,330	do.....	Both headings, January, 1910.	February, 1912.
Tunnel 17B.....	9,220	do.....	Both headings, March, 1910.	(e)
Elizabeth division:				
Elizabeth Lake Tunnel.	26,870	do.....	South heading, Oct. 5, 1907; north heading, Nov. 1, 1907.	Feb. 28, 1911.

* Dec. 1, 1911, north heading had been driven 3,800 feet; south heading, 3,634 feet; completed in 1912.

AMERICAN TUNNELS DESCRIBED IN ENGINEERING MAGAZINES.

The following data are comparable with those above, and give practically similar information concerning certain American tunnels that were not examined in the field but are rather fully described in engineering periodicals. Although the information contained in the various accounts is, perhaps, somewhat less complete than similar data obtained at other tunnels actually visited, nevertheless it is generally sufficient for each tunnel to convey a good idea of the main features of the work done.

TABLE 4.—Comparative data of American tunnels described in engineering magazines.

Name of tunnel.	Location.	Operator.	Purpose.	Shape of cross section.	Size.	Length.	Character of rock penetrated.
Buffalo Water Works ^a	Buffalo, N. Y.	Department of Public Works, Buffalo.	Water supply	Nearly rectangular.	15 feet wide, 15½ feet high, as excavated.	<i>Feet</i> , 6,575	Limestone.
Chippeta Adit ^b	Ouray, Colo.	United Railways Co.	Mining.	Square.	7½ feet by 7½ feet.	2,000	Hard.
Cornelius Gap ^c	Near Portland, Oreg.	City Water Works	Electric railway	Arched roof.	17½ feet wide, 22½ feet high (above ties).	4,100	Basalt.
Fort William ^d	Fort William, Ontario	New York Central & Hudson River R. R. Co.	Water supply	do.	5 feet wide, 6½ feet high.	4,820	Do.
Grand Central Terminal sewer ^f	Grand Central Station to East River, under Forty-sixth Street, New York City.	Red Mountain Railroad, Mining & Smelting Co., Bunker Hill & Sullivan Mining Co., City of Chicago.	Storm-water discharge.	Circular.	6 feet diameter inside of lining.	3,000	Hard gneiss.
Joker (drainage) ^e	Red Mountain, Colo.	Shoshone County, Idaho.	Mine drainage and development.	Rectangular.	12 feet wide, 11 feet high, as excavated.	5,065	Very heavy, requiring much timber.
Kellogg ^g	Shoshone County, Idaho.	do.	Mine development.	Arched roof.	9 feet wide, 11 feet high.	9,000	Quartzite.
Northwest ^h	Chicago, Ill.	City of Chicago.	Water supply	Horseshoe.	Area equivalent to 14-foot circle.	21,180	Granite and breccia.
Ophelia ⁱ	Cripple Creek, Colo.	do.	Mine drainage and development.	Rectangular.	9 feet by 9 feet.	8,500	Shale, sandstone, and a 3-foot bed of soft coal.
Second Raton Hill ^j	Raton Pass, Colo.-N. Mex.	Atchison, Topeka & Santa Fe Ry.	Railway.	Horseshoe.	22 feet wide, 29 feet high, as excavated.	2,780	Crystalline limestone
"Spiral" ^k	Selkirk Mountains, British Columbia.	Canadian Pacific Ry.	do.	Arched roof.	22 feet wide, 27 feet high, as excavated.	(*)	

^a Eng. Rec., June 25, 1910, p. 802.^b Min. and Sci. Press, July 11, 1906, p. 60.^c Approximate.^d Eng. News, June 29, 1911, p. 783.^e Eng. and Contrng., May 25, 1910, p. 472.^f Eng. Rec., Apr. 11, 1906, p. 496.^g Mines and Minerals, May, 1907, p. 470.^h Mines and Minerals, October, 1901, p. 122.ⁱ Eng. Rec., Aug. 7, 1909, p. 144.^j Mine and Quarry, May, 1907, pp. 118-122.^k Eng. Rec., Apr. 4, 1903, p. 461.^l Eng. News, Nov. 10, 1910, p. 512; Comp. Air Mag., February, 1911, p. 9831.^m Tunnel No. 1, 3,200 feet; Tunnel No. 2, 2,950 feet. Both curved; 573 feet radius.

CAUSES AND PREVENTION OF TUNNEL ACCIDENTS.

Data collected by the Bureau of Mines show that an average of nearly 4 men for each 1,000 employed in and about the metal mines of the United States were killed during the year 1911, as compared with 3.8 per 1,000 in coal mining during the same period. Although complete figures for accidents in tunnel driving can not be obtained, a study of such data as could be collected indicates that the number of deaths per year per 1,000 men employed has been somewhat greater than the above figures, the result obtained by averaging data extending over periods of 1 to 10 years for 16 representative tunnels being 4.7 deaths per year per 1,000 men employed. In addition to the men killed outright by accidents in tunnel work, nearly 3 times as many more have been seriously injured or perhaps maimed for life, and almost 13 times as many slightly injured. By far the largest part of these deaths and injuries was caused by falling ore or rock from the roof or walls of the tunnels, but explosives, haulage, electricity, and other causes have each contributed their quota of casualties.

Are these accidents preventable? Not entirely, because some elements of danger are inherent in the work of driving tunnels; such, for example, as the danger from some unforeseen falls of roof, from the derailment of tunnel cars, or the risk involved in handling even the least dangerous explosives by the most approved methods. But it is equally true that much of the present mortality and injury is the result of ignorance or gross carelessness, and can be avoided. When, for instance, a man sees fit to thaw frozen dynamite in a frying pan or by a candle flame, there is nothing accidental about the explosion that ensues, except, possibly, the fact that a man so ignorant or reckless should have been intrusted with so dangerous a substance. Nor is the responsibility for accidents all on the part of the miner. The manager and his representatives are in many cases either ignorant of the precautions that should be taken for the safety of the men or most negligent in seeing that they are properly and consistently carried out. The following discussion of causes of tunnel accidents is presented in the hope that, by bringing these matters once more squarely to the attention of the men interested, much of the needless death and suffering may be prevented.

CAUSES OF ACCIDENTS.

FALLS OF ROOF.

There are many causes that combine to make falls of rock from the roof by far the greatest source of danger in tunnel work, but perhaps the chief of these is the common practice of greatly overloading the

holes with explosives. Extremely heavy charges shatter and crack rock that would ordinarily stand without any danger of falling, and render it extremely dangerous to the men working underneath. Of course it is essential to efficient work in tunnel driving that the blast should completely "break bottom" without any necessity for a second loading and firing; still every foreman and superintendent should see that the smallest amount of dynamite that will do the required work is employed in the holes near the roof. Economy of explosive demands this, all other considerations aside; but the dangers, also, of the heavier charges should be thoroughly appreciated by the superintendent and, when such charges seem imperative, extra vigilance should be exercised and extra precautions taken for the safety of the men.

Another prolific source of accident is the fact that men sometimes return to the tunnel face, after shooting a round, without thoroughly testing the roof just exposed by the blast. It should be the duty of every man employed in the tunnel to examine the roof under which he must work, and especially in that part of the tunnel newly exposed after shooting; the foreman, upon reaching the heading after the blast, should at once detail one or two men (or as many as prove necessary) to clean down thoroughly all the loose pieces of overhead rock. Fortunately, this is done regularly at all well-organized tunnels and it is a practice that can not be too highly recommended for universal adoption.

It must be admitted that from a roof declared by experienced men to be sound, a large block may suddenly and without warning crash into the tunnel. This occurrence will undoubtedly be claimed to have been purely accidental; yet even the danger from such a block (which perhaps was perfectly solid when first exposed, but became loosened by the concussion of subsequent blasting) is, in many cases, overlooked because of the lack of illumination in which all tunnel work must be done, and may be discovered in time if there is a systematic and regular examination of the entire roof of the tunnel. Some one has pointedly observed, "The fall of a slab of rock weighing anything less than 1 ton should at once be charged to carelessness."

It should be said in this connection that the "sound" of the roof is not a proper criterion of its safety, because there are numerous cases on record where the sound of the roof was satisfactory and indicated rock that seemed solid even to experienced men, although a big block or boulder was actually loose. The better method of testing the roof—one used by many large mining companies and recommended by the Bureau of Mines^a—is to strike it with a pick or a heavy stick, at the same time touching the doubtful piece with the free hand. If any vibration is felt the rock is unsafe and should

^a Rice, G. S., *Accidents from falls of roof and coal*: *Miners' Circular* 9, 1912, p. 8.

be taken down or supported at once. If the roof is too high to reach with the hand, a stick should be held against the doubtful piece while it is being struck, and if it is loose the vibration can be felt through the stick.

Prompt and adequate timbering is extremely important. But timbering is a laborious process and it either takes the men of the tunnel crew from their regular work, or it requires extra men. Extra men, however, add to the confusion in the heading and, as their work is done simultaneously with the other work of the tunnel, it seriously hinders either the drillers or the shovelers, or both. So it has become recognized among tunnel men that in most cases timbering seriously impedes the progress of driving, and therefore, although it may be well understood that the roof is dangerous, there is almost always a tendency on the part of those responsible to delay timbering as long as possible. Perhaps the American willingness to "take a chance"—a trait particularly noticeable in the Western States—may be a contributing cause; but the fact remains that the work of timbering is too often delayed until a so-called "accident" brings the necessity forcibly and unavoidably to the front. It is impossible to urge too strongly that all necessary timbering be done promptly, that it can not be done too soon, and that any delay seriously jeopardizes the lives and limbs of the men who have to work under a roof improperly supported.

It is true that in many tunnels the weight of the roof or pressure against the walls has been too great even for the strongest and heaviest timbering, and although such breakage can not always be prevented, it may often be alleviated by means discussed in the section on timbering. The important consideration in these cases as regards safety is the fact that actual failure of the timbers and caving of supported ground rarely comes without warning. Either the timbers will at least be bent appreciably before they break, or, as is usually the case, they will crack and splinter and so give unmistakable warning to the miner that the time is approaching when they will collapse. With such warning any subsequent accident is chargeable to carelessness or negligence in heeding the danger signal. It may be said in this connection that, other things being equal, timber that has a fiber that will split, crack, or splinter out, rather than that which has a fiber that will break off short under a transverse strain, is on this account more desirable for such work.

Falls of rock are also caused by cars becoming derailed and knocking out the supporting timbers under a heavy or loose part of the roof, allowing the roof to fall and kill or injure any men who happen to be underneath. Such accidents are in many cases unavoidable because of the difficulty of preventing derailments. Owing to the lack of illumination, it is usually impossible to see whether the track

ahead is clear, and it is therefore necessary to run somewhat blindly and assume that nothing has fallen upon the track since the previous trip; and the mere work of keeping the roadbed of a tunnel track in such shape that its unevenness would no longer cause the cars to jump off would be enormous. The only way, therefore, to lessen these accidents (which fortunately are not so numerous as from other causes) is to keep the track in as good condition as possible, and to use all reasonable watchfulness and caution in tramming, and to avoid in particular running trips at a high speed over bad track.

USE OF EXPLOSIVES.

Next in importance as a cause of injury in tunnel work is the careless, reckless, improper, or ignorant use (or rather misuse) of explosives. Such accidents are of various kinds, the most frequent being those arising from handling, storing, and thawing dynamite, from premature blasts, from misfires, or from poisoning by gases from explosives. The subject of the proper ways to handle, store, and thaw dynamite is treated at some length in the chapter on blasting, but as it is impossible to place too much emphasis upon the necessity for care and caution in the use of explosives, a recital here of the precautions to be taken is well warranted.

PRECAUTIONS AS TO HANDLING.

Don't forget the nature of explosives, but remember that with proper care they can be handled with comparative safety.

Don't smoke while handling explosives and don't handle explosives near an open light.

Don't shoot into explosives with a rifle or pistol, either in or out of a magazine.

Don't attempt to manufacture any kind of an explosive except under the supervision and direction of a trustworthy person who is skilled in the art. Many serious accidents, which have destroyed lives or inflicted injury on persons and property, have been caused by such attempts.

Don't carry blasting caps or electric detonators in the clothing.

Don't tap or otherwise investigate a blasting cap or electric detonator.

Don't attempt to take blasting caps from the box by inserting a wire, nail, or other sharp instrument.

Don't try to withdraw the wires from an electric detonator.

PRECAUTIONS AS TO STORING.

Don't leave explosives in a wet or damp place. They should be kept in a suitable, dry place, under lock and key, and where children or irresponsible persons can not get at them.

Don't store dynamite so that the cartridges are on end, as this position increases the danger of nitroglycerin leaking.

Don't store or handle explosives near a residence.

Don't open packages of explosives in a magazine.

Don't open dynamite boxes with a nail puller, or powder cans with a pickax.

Don't store or transport detonators and explosives together.

Don't keep electric detonators, blasting machines, or blasting caps in a damp place.

Don't allow priming (the placing of a blasting cap or electric detonator in dynamite) to be done in a thawing house or magazine.

PRECAUTIONS AS TO THAWING.

Don't use frozen or chilled explosives. Most dynamite freezes at a temperature between 45° F. and 50° F.

Don't thaw dynamite on heated stoves, rocks, sand, bricks or metal, or in an oven, and don't thaw dynamite in front of, near, or over a steam boiler or fire of any kind.

Don't take dynamite into or near a blacksmith shop or near a forge.

Don't put dynamite on shelves or anything else directly over steam or hot-water pipes, or other heated metal surface.

Don't cut or break a dynamite cartridge while it is frozen, and don't rub a cartridge of dynamite in the hands to complete thawing.

Don't heat a thawing house with pipes containing steam under pressure.

Don't place a "hot-water thawer" over a fire, and never put dynamite directly into hot water or allow it to come in contact with steam.

LOADING PRECAUTIONS.

Don't allow thawed dynamite to remain exposed to low temperature before using it. If it freezes before it is used, it must be thawed again.

Don't fasten a blasting cap to the fuse with the teeth or flatten the cap with a knife; use a cap crimper. The ordinary cap contains enough fulminate of mercury to blow a man's head or hand to pieces.

Don't "lace" fuse through dynamite cartridges. This practice is frequently responsible for the burning of the charge.

Don't explode a charge to chamber a hole and then immediately reload it, as the bore hole will be hot and the second charge may explode prematurely.

Don't force a primer into a bore hole.

Don't do tamping with iron or steel bars or tools. Use only a wooden tamping stick with no metal parts.

Don't handle fuse carelessly in cold weather, for when it is cold it is stiff and breaks easily.

Don't cut the fuse short to save time. Such economy is dangerous.

Don't worry along with old broken leading wire or connecting wire. A new supply will not cost much and will pay for itself many times over.

FIRING PRECAUTIONS.

Don't explode a charge before every one is well beyond the danger line and protected from flying débris. Protect the supply of explosives also from the flying pieces.

Don't hurry in seeking an explanation for the failure of a charge to explode.

Don't drill, bore, or pick out a charge that has failed to explode. Drill and charge another bore hole at least 2 feet from the missed one.

PREMATURE EXPLOSIONS.

It is often difficult to determine just what were the causes of premature explosion, because the persons responsible for the explosion rarely survive to tell the tale and even eyewitnesses are scarce; but carelessness in handling the dynamite in the heading is no doubt the most potent factor. In many cases the so-called accident does not result from the first instance of carelessness or recklessness, but is the disastrous climax of a series of practices that have become habitual; hence persons knowing the common disregard for dynamite on the part of the men who handled it and were killed are able to draw accurate conclusions as to the probable cause of the "accident." As an example might be cited the case of two men who were accustomed to throw sticks of dynamite to each other along the tunnel, over distances of 15 or 20 feet, especially if visitors with "nerves" were present. But even at other times, perhaps because of long familiarity with dynamite and hence a contempt or disregard of its true dangerousness, the sticks were thrown to one another rather than carried the few intervening feet. However, the practice as far as these two personally were concerned, was finally stopped by a disastrous explosion in which they were blown almost to atoms. The subsequent appearance of the tunnel indicated that the explosion was caused by the detonation of a stick falling near the full supply for the entire round.

Another cause of premature explosions is the practice of carrying dynamite to the face of the tunnel in a box or sack and dropping it rather roughly to the ground at the end of the journey. This contempt is also bred, no doubt, by familiarity. It is true that oftentimes gelatin dynamite is not as sensitive to direct shocks as one might imagine, and that many times it will stand very rough usage without detonation; but in other cases, and there are very many of them on record, serious explosions have ensued as a result of inex-

cusable carelessness in handling. It is neither safe nor advisable to rely in any degree whatsoever upon the "inertness" of dynamite. Nor is it possible to condemn too strongly the practice of carrying detonators or primers (sticks of dynamite containing a detonator and a fuse) in the same bundle with the rest of a supply of explosive for a round. The detonators should always be brought in separately and should under no circumstances be placed in the same box or even near together in the heading. Many serious accidents have resulted through disregard of this rule.

A certain risk must always attend the loading of a bore hole with dynamite, especially during the insertion of the primer, but much of the danger that often needlessly accompanies this work can be minimized or avoided by proper care. Efficiency of course demands that there shall be no air spaces in the charge of explosive when it is finally ready for detonation; hence the dynamite must be rammed down so that it fills all the unequal spaces in the bore hole; but tamping should always be done by pressure rather than impact. Never use a tamping bar as if it were a javelin. But even in pressing down the charge, great care must be taken that too much force is not employed, especially if a cartridge seems to stick in a hole; for should it become suddenly loosened the miner might not be able to recover himself in time to prevent its being rammed hard against the bottom with disastrous results. Anything more than light pressure should never be given the primer and under no circumstances should it or the succeeding cartridge be struck a blow with the rod.

Irregularity in the rate at which fuse burns is also a cause of premature explosions. Different makes and brands of fuse burn at different rates, and a miner accustomed to a slow-burning fuse will perhaps not realize the necessity of cutting the faster fuse longer, so that he may have time enough to reach a place of safety before the detonation takes place. There are several causes of variations in the burning rate even of the same brand of fuse. For example, experiments conducted by the Bureau of Mines* show that mere confinement in a closed vessel is sufficient to cause a fuse to burn three or four times faster than its normal rate. It is true that under ordinary conditions of mining, variations of this magnitude are not apt to be reached, but irregularities of 20 or even 30 per cent are quite possible and in long bore holes in which a quantity of tamping is used, especially of a type impervious to the escape of the gases (such as closely packed wet clay), the variation may be much greater. Therefore, with such tamping, the rate of burning may be increased to a dangerous extent, unless due allowance be made for the extra speed. But even more important is the effect produced by mechanical injury,

* Snelling, W. O., and Cope, W. C., The rate of burning of fuse as influenced by temperature and pressure: Technical Paper 6, Bureau of Mines, 1912, 28 pp.

which is more apt to be a common occurrence. Mere bending of fuse (if it is in proper condition for use), such as might result from coiling it near the collar of the hole to prevent its being struck by flying rock from other blasts, or even placing it with some force within the hole, has little if any effect upon the rate of burning; but abrasion, blows, or too great pressure produces serious variations in this rate and in some cases may even cause fuse to burn almost instantaneously. It is therefore essential that none but fuse in good condition ever be brought into the heading, and that care be taken while it is there to see that it is not injured by rocks or tools falling on it, and that it is not abraded or otherwise injured with the tamping bar while the hole is being loaded.

Mention should be made of the seemingly obvious danger of re-loading a bore hole before it has had time to cool off sufficiently from a previous blast. In tunnel work this danger occurs in connection with the "guns"—the ends of holes that have not broken to the bottom with the first explosion.

MISFIRES.

Many deaths and injuries are caused by the subsequent detonation of a charge of dynamite that failed to explode at the proper time. Such misfires do not, however, cause accidents unless the charge is detonated unexpectedly. Sometimes this happens by drilling into it during preparations for the next round, or by striking it in the muck pile, where it has been thrown by the blast from a neighboring hole, or perhaps by the sudden explosion of a delayed shot from a fuse that has long been smouldering.

Many misfires can be traced directly to some injury to the fuse. The insertion of the primer into the hole, fuse end first, often causes fuse to crack at the sharp bend thus made; the danger of such cracking is especially great when the fuse is cold or the hole is full of cold water. Sudden and rough uncoiling of the fuse in cold weather will usually cause it to break. Obviously, therefore, cold fuse should not be bent, twisted, or roughly handled. It is claimed by some persons that misfires are caused through fuse being cut off ahead of the fire by the explosion of a neighboring hole, so that the charge fails to explode. There is some question whether this really happens or not; but, if it does, it is a pretty strong argument that the hole was misplaced, for if a hole is properly placed, only in rare instances, if ever, will enough of it be shot away to cut off the fuse ahead of the fire. It is also claimed, and with somewhat more reason, that the fuse is apt to be torn out by flying pieces of rock from the explosion of other holes, but this result can be largely obviated if the fuse is properly coiled close to the mouth of the hole before it is "spit."

The failure of a fuse properly to ignite a detonator is often the result of improper storage. When the asphalt waterproofing composition used in some fuses gets too hot it becomes viscid and agglomerates the powder grains in the core of the fuse and thus delays, and in some cases actually prevents, the fuse from burning. Experiments conducted by the Bureau of Mines* indicate that prolonged exposure at a temperature of 60° C. is sufficient to cause a marked retardation in the rate of burning of fuse. It follows, therefore, that fuse should not be stored near boilers, steam pipes, or other sources of heat, where the temperature is apt to be high. Cold is likewise deleterious, for it renders the asphalt composition brittle and liable to crack, and these cracks either decrease the rate of burning by permitting the gas from the powder core to escape more readily than usual, or, if they are large enough, they may stop the travel of the fire entirely. The fuse should be carefully protected from moisture during storage for, with waterproof fuse of the type almost universally employed in tunneling, if the dampness once gets into the powder train its removal is difficult. As the fuse burns, the moisture is driven ahead of the fire in the form of steam and even if it does not accumulate in sufficient quantity to quench the fire in the fuse, enough of it may be driven into the detonator to prevent ignition and thus cause a misfire.

Many misfires originate from improperly prepared primers. Before the fuse is inserted into the detonator, an inch or two should be cut off and thrown away, for gunpowder (which forms the core of the fuse) is somewhat hygroscopic, and the end of the fuse may have gathered moisture enough to quench the burning powder or prevent the ignition of the cap. This cut should be made with a sharp-cutting tool, squarely across the fuse, for if made diagonally the point may curl over the end of the fuse when inserted in the detonator and thus prevent the spit of the powder train from reaching the detonating composition in the cap. Care should also be taken that the powder grains in the end of the fuse do not leak out after the fuse has been cut, for this would tend to weaken the force of the spit into the detonator and might prevent its ignition. The open end of the cap should be carefully crimped around the fuse with a proper crimping tool, so that it will be tight enough to hold the detonator and the fuse together and keep out moisture, but the crimping should not be tight enough to cut off the powder train in the fuse. This is particularly liable to happen with a narrow crimping tool that presses a narrow groove in the detonator and the underlying fuse. There are tools on the market that have a crimping face of at

* Snelling, W. O., and Cope, W. C., The rate of burning of fuse as influenced by temperature and pressure, Technical Paper 6: Bureau of Mines, 1912, p. 19.

least a quarter of an inch, and the extra price of these tools would be no more than the cost of the explosive wasted by a single misfire—to say nothing of the loss of life that might arise therefrom. It is, of course, obvious that the teeth or a knife should never be used for crimping, for, as previously stated, there is enough explosive in an ordinary detonator to blow a man's head or hand to pieces. After it is crimped, the detonator should be buried in the end of the stick of dynamite, with its axis parallel to that of the stick, and the top of the detonator should be flush with the top of the dynamite. For if the cap is buried deeper, the explosive is liable to become ignited from the side spitting of the fuse before it is properly exploded by the detonator, a result that not only destroys the efficiency of the explosive, but causes a larger amount of gases, especially those most dangerous to the men who must breathe them. It is also important to use a detonator of sufficient strength. Although 3X blasting caps were considered strong enough for "straight" nitroglycerin dynamite, the less sensitive gelatin dynamite requires a much stronger detonator to explode it properly. For this reason nothing weaker than 5X caps should ever be used with gelatin dynamite, and the universal experience is that better results have been obtained when a change has been made to even stronger detonators. These insure the complete detonation of the explosive and thus produce only a minimum amount of dangerous gases.

It is very difficult to count the explosions during blasting and be sure that the charges have all been detonated, so it is not always possible to determine whether there has been a misfire. For this reason the face, or as much of it as is not covered by the débris resulting from the blast, should be inspected for evidences of missed holes, and it should be carefully watched during the removal of the muck. If a missed hole is discovered, under no circumstances should an attempt be made to pick out the material. If no tamping has been used, a stick of dynamite containing a detonator should be inserted in the hole and exploded. If tamping has been employed, another hole should be drilled and blasted at least 2 feet from the missed one. In picking down the muck pile the pick should be handled as if it were a hoe and not like a sledge hammer; that is, the material should be pulled or scraped down and never struck violently with the point of the pick. In this way, should there happen to be a piece of unexploded dynamite in the débris, there is much less danger of its exploding. The importance of this precaution can not be too strongly emphasized. Should a piece of dynamite be discovered in the muck, it should be removed carefully and handed to the foreman who should at once take it to a safe place, and extreme care should be used if a piece of fuse accompanies it or is discovered near it, for this would indicate that an unexploded

detonator may possibly still be inside of the stick of dynamite, the danger of which is obvious. Under no circumstances should a new hole be started in the remnants of a hole that has ever held dynamite; for although the inference is always, of course, that the dynamite has been detonated, still there remains a chance that detonation has not occurred—a chance not as slight as ordinarily might be supposed, to judge from the number of accidents traceable to this source. And even if a rod be used to test the hole, it might encounter a small obstruction and by thus seeming to show the bottom of the hole fail to reveal the dynamite beneath.

GASES FROM EXPLOSIVES.

Poisoning from the gases produced by explosives is common in tunnel work. The ailment is familiar to most miners; in its mild form it is usually called "powder headache" and produces little more than temporary inconvenience, but in severe cases it has been known to produce death within a very short time. In the section on blasting it is explained that the harmful gases resulting from the complete detonation of dynamite under normal conditions are usually carbon dioxide and carbon monoxide; that although carbon dioxide will not support respiration, and when present in sufficient quantities may cause unconsciousness and even death, it has no very injurious effects when sufficiently diluted; that carbon monoxide is exceedingly dangerous and even small amounts of it may prove fatal if breathed for a sufficient length of time. This gas probably causes the familiar symptoms after a dose of "powder smoke." By reference to the table on page 153, it will be seen that gelatin dynamite, the explosive almost universally used in tunnel work, under proper conditions generates comparatively little of the more dangerous gas. Experiments conducted by the Bureau of Mines^a indicate that even this can be obviated by a slight modification in the chemical composition of the gelatin dynamite. But when even such a dynamite is not completely detonated (either through the use of too weak a detonator or any other cause), and especially when it burns rather than explodes, a much greater volume of monoxide is formed, and in addition there are a number of other harmful gases developed, including the dangerous peroxide of nitrogen. It is therefore essential that the detonators employed be strong enough to explode the dynamite completely, and that every precaution be taken to prevent the dynamite from taking fire through the side spitting of the fuse or in any other manner.

^a Hall, Clarence, and Howell, S. P., The selection of explosives used in engineering and mining operations: Bull. 48, Bureau of Mines, 1913, 50 pp., 3 pls., 7 figs.

The deadliness of the gases resulting from explosives improperly detonated may be illustrated by describing an accident that is known to have cost 9 lives. A study of the attendant circumstances, as described to the writers, indicates that the explosive, or at least a large part of it, must have burned rather than detonated. Gelatin dynamite was employed and the charge was even smaller than previous blasts of which the men had inhaled the fumes without serious effects, but in this case the fumes are described by the men as being brownish yellow rather than the usual grayish or bluish white. After igniting the blast the men retired about 500 feet to wait for the smoke to clear, and while they were waiting the smoke drifted slowly over them and then, owing to some change in the current, drifted slowly back again. The men soon felt the usual symptoms of carbon monoxide poisoning—slight choking, nausea, profuse perspiration, and headache—but they all revived upon reaching the open air about an hour and a half after the blast was fired. Within a short time, however (and in one case before the man could walk to the bunk house), the men began to cough up bloody mucus and to exhibit other symptoms of nitrogen peroxide poisoning, and in less than three days 9 of the 13 men who had been in the tunnel and exposed to the fumes had died. The 4 who escaped were either not exposed to the gas for the full time, or else found some other source of air supply which served partly to dilute the gases; but some of these men as well as those who went in with the motor to bring the men out were ill for days and even months after the catastrophe.*

It is the opinion of physicians who have studied the matter that many swift deaths among miners, formerly diagnosed as pneumonia, may really have been caused by the inhalation of gases from burning dynamite.

GASES FROM OTHER SOURCES.

Although any carbon monoxide encountered in tunnel work is liable to be a result of the use of the dynamite, there have been cases where this dangerous gas has been generated by the combustion of oil and grease in the air receiver and transmitted to the heading by the compressed-air pipe. The causes of such combustion are fully discussed in the section on air compressors, but mention is here made that the ignition of accumulated oil and grease is generally due to faulty valves in the compressor. These permit warm compressed air to leak back into the cylinder; this air upon being recompressed becomes still hotter, so that after a time the temperature of

* A full discussion of the customary symptoms that accompany poisoning from nitrogen peroxide and carbon monoxide and their comparison with the symptoms exhibited by the men is contained in the April, 1911, number of *Colorado Medicine*: "An Unusual Powder-Smoke Fatality," by Carl Johnson.

the air in the receiver may be far higher than the ignition point of the lubricant employed. If an explosion does not then ensue, the oil on the sides and bottom of the receiver will burn and produce carbon dioxide or carbon monoxide, either of which jeopardizes the safety of the miner in the heading. It is therefore necessary to inspect the valves of the compressor regularly; moreover, dependence should never be placed on the compressed-air line for tunnel ventilation.

There are several tunnels in which bodies of gas have been encountered, the gases most frequently found being carbon dioxide and hydrocarbon gases. The former is, of course, chiefly dangerous because of the possibility of the men being suffocated, but this can be largely obviated by proper ventilation. In one of the tunnels of the Los Angeles aqueduct, flows of carbon dioxide were encountered in a series of crevices across a zone about 150 feet wide. In order to make it possible for the men to work in the tunnel this zone, including 300 feet on either side, was tightly sealed with concrete; in addition it was found necessary to leave in the center of the gas zone back of the concrete an annular space to which an exhaust "blower" was connected that constantly drew off the gas during the driving of the tunnel, while an additional blower forced fresh air in to the men. If either of these machines stopped the men had to get out of the tunnel as fast as possible, but as long as the machines kept running the air was sufficiently pure.

The chief danger from hydrocarbon gases lies in their explosibility, but they are so commonly encountered in coal mining that precautions to be taken in their presence are fairly well known. However, a rather unique although highly dangerous method of dealing with them was employed in one of the tunnels examined by the writers, and is well worth describing.

The gas was encountered in a zone approximately 2,300 feet in extent, through about 500 feet of which oil could be distilled from the rocks, although there was no seepage. The gas was highly explosive, and had an odor of kerosene or gasoline rather than of crude petroleum. The largest quantities of it came into the tunnel immediately after blasting, and the maximum accumulation was approximately 30,000 cubic feet. There did not appear to have been any particular seepages in the gaseous zone, but rather there was always an unknown quantity ahead of the work. As the gas was highly explosive extra precautions had to be taken for the safety of the men at work. The mere requirement of safety lamps in the tunnel was not considered sufficient, because the very nature of the rock was such as to cause dangerous sparks from a pick or from the starting of a drill hole, which it was thought would be sufficient to ignite the gas and produce an explosion. The expedient adopted

was to explode the accumulation after each blast and to burn any new gas as fast as it appeared in the tunnel during the remainder of the work.

For this purpose the tunnel was wired from the portal to the heading with a 550-volt circuit, into which there were introduced at intervals of about 200 feet throughout the entire gas-bearing section a number of arcing devices. Any ordinary street arc lamp could have been adapted for this work, provided that the carbons were not exposed for more than 2 inches; otherwise the concussion from ordinary blasting, as well as from the gas explosions, would have broken them. The use of one soft and one hard carbon was found to give the best results. The system was operated as follows:

Immediately after blasting, a fire boss and his helper took charge of the tunnel. After waiting 30 minutes after the blast had been fired they turned a current of electricity through the arc line by means of a switch at the portal. The arcs were purposely placed in series in order to make certain that if any one of them burned they would all burn; an ammeter was placed at the control switch to show whether they had lighted. If the arcs did light, an explosion generally ensued, sometimes a severe one. But whether or not there was an explosion the switch was always opened for 15 minutes and then closed a second time as an added precaution, although a second explosion never resulted. When the line was dead once more two men carrying safety lamps proceeded to a protected station approximately halfway to the heading, where they again sent a current through the arcs. A few explosions resulted from this practice, but they were unusual rather than customary. After having made this test the fire boss and his helper proceeded to the heading, testing the entire tunnel for gas by means of the safety lamps they carried. They would ordinarily find in the heading an accumulation of gas extending back a distance of 125 to 150 feet, because the nearest arc could not be placed much nearer to the heading than 150 feet on account of the danger of the carbons being broken by the concussion from the blasting. The fire boss would then take an arc kept 150 feet from the face and attached to the circuit by an armored cable and place it over the muck pile; the two men would return again to the midway station and once more close the circuit and ignite the remaining gas. Then, and then only, with all the arcs burning, they would return to the heading and place torches as near the roof as possible at intervals of about 150 feet throughout the gaseous section. The torches were lighted from the arcs, and the men were not permitted to light them in any other way, or, indeed, to carry into the tunnel any other means of lighting them. By this time all the seepages that were strong enough to support a steady flame would have been lighted and would be burning, and the

gas that came from pockets that could not sustain a flame would be ignited by the torches before it could accumulate in any quantity.

The fire crew then returned to the mid-station, where they extinguished a red light and lighted a white one, indicating that the tunnel was safe for the incoming crew, for no one but these two men were allowed in the tunnel beyond this point unless the red light was out and a particular white one burning, in order to obviate danger through any accidental extinguishing of the red light without the knowledge of the fire crew and before the tunnel was safe. The fire crew was allowed four hours for this work, although ordinarily that length of time was not required.

The working crew upon reaching the heading ordinarily found the muck pile too hot to be handled, if, indeed, it was not actually in flames, for it burned usually for one-half to two hours after each blast, and once at least it burned for 14 hours. After it had been cooled sufficiently by streams of both air and water, the machines were set up and the round of holes drilled in the regular manner. Any gas that developed during the drilling of a hole was lighted as soon as the hole had been completed, and if sufficiently strong to support a flame it would burn until the end of the shift. At one time as many as 6 out of 8 holes on the top round were burning like blow-torches, giving flames 6 to 18 inches in length. When the round had been finished the holes had to be cooled before loading. This was accomplished by turning water and air lines through ordinary blow-pipes, both into the holes and over the face of the tunnel. The flames were, of course, extinguished by this process, and as soon as the gas had accumulated in the tunnel sufficiently to become apparent in a safety lamp placed near the roof about 30 feet from the heading, it was ignited by a torch and the resulting flames were at once put out again by air and water. This process was continued until the holes were cool, when they were at once loaded as rapidly as possible and fired, the fuses being always lighted from near the bottom of the tunnel.

Although the fact that there were no accidents in driving through the gas-bearing zone after the installation of the "safety arcs" shows that this system was efficacious in this particular instance, it is not one that can be recommended unqualifiedly for general use. In the opinion of engineers who have made a special study of the question of safety in mining, the use of anything but safety lamps or their equivalent in mines or tunnels where explosive gases are known to exist is never without risk, whereas the practice of burning the gases as fast as they make their appearance is in itself extremely hazardous. Indeed, the fact that no disastrous explosion occurred under this system seemed to them remarkable. Moreover, it is obvious that long delays were necessary before the men could start to work, and

even after they had reached the heading the heat must have greatly decreased their possible efficiency. A less dangerous method of handling the gas, and one that would probably prove more economical in the end, would be the installation of a ventilating system large enough to dilute to harmlessness several times the amount of gases ordinarily encountered. Safety lamps only should be allowed in the tunnel and all blasts should be fired by electricity.

HAULAGE.

A large proportion of the injuries attributed to tramming is caused by the practice of riding on the cars, especially loaded ones. When riding on the top of a full trip, a man is always in danger of a serious injury at every low place in the roof, and if he is riding between the cars (or any place but the rear end), he is liable to be jarred from his foothold and dragged under the cars, and in case of derailment he has little chance of escape. The risk of derailment is unavoidable in tunnel work, partly because of the insufficient illumination under which tramming is generally carried on, and partly because of the difficulty of keeping the roadbed in good condition or the track clear of small obstructions. Even when riding on empty cars there is serious risk whenever the miner sits on the ends or sides and allows his feet to hang over; the safest way is to sit inside of the car and to crouch low enough to avoid being struck by any jutting place in the roof. The arms and hands should be kept inside of the car to avoid the possibility of being caught between the car and the wall at a tight place. The driver or "mule skinner" is usually compelled to ride on a loaded trip and sometimes at the front end of the train in order to be near the animal he is driving; the extra hazard of this position should be fully realized and extra precautions taken. The dangerous practice observed on the part of some drivers, of riding with one foot on the bumper and the other on the chain by which the mule pulls the trip, is very obvious and can not be too strongly condemned. This act should be made sufficient cause for instant dismissal. It ought not to be necessary to mention the danger of attempting to jump on or off a moving trip of cars, because the chances in such a case of a man missing his footing and being caught or dragged under the cars, or of breaking an ankle or leg in the uncertain light, should be so clearly seen that no one ought to consider the risk worth taking; but the number of injuries arising from this cause shows only too well that this precaution is habitually disregarded.

Great care is necessary during the operation of placing a derailed car back upon the track. It is very easy for a miner to strain or otherwise injure himself if he attempts to do this without getting

some one to assist him. Also in handling a derailed car that is full of rock there is danger of the block or crowbar slipping and allowing the car to drop suddenly on the miner's foot or hand, if indeed it does not topple over completely and crush him against the side of the tunnel.

Failure to allow sufficient room to a passing trip of cars is also a frequent source of injury. Before going into a strange tunnel the miner, if he is not accompanied by some one familiar with the tunnel, should always ascertain upon which side of the track there is the most room, and in meeting a passing trip should always give the animal pulling it all the space possible, so as to avoid being tramped on or kicked, or being caught between the cars and the walls of the tunnel. It is also advisable to hide any light when meeting a horse or mule, for there are some animals that are especially afraid of the high-powered acetylene lamps that are coming to be used almost entirely in tunnel work. If the animal balks when coming toward a light a serious mixup may occur, as the cars behind can not always be stopped at once. In a tunnel, as on the surface, attention should always be given the heels of animals whether moving or at rest, and it is best to speak to animals when approaching them from behind, for many serious injuries have been caused by passing too close to nervous animals without warning. When turning a horse or mule around in a heading, the driver should watch carefully to see that he is not stepped on; inane as this advice sounds, many really serious accidents have resulted from just this simple cause.

ELECTRICITY.

An examination of reports of electrical accidents in tunnel work shows that in most cases the shocks were caused by the trolley wire. This is not surprising when one considers the many factors that unite to make an electrically charged wire especially dangerous underground. The earth is almost always used to complete the return circuit and, therefore, if the miner inadvertently touches any part of an electrical apparatus that is charged with current, and if he is not well insulated from the ground, he will certainly get a shock, the intensity of which will depend on the voltage or pressure of the electric current and the incompleteness of his insulation from the earth. Some trolley wires carrying a current as high as 600 volts have no insulating or protecting covering whatsoever and most of them are without a guard or shield of any sort, although they are sometimes placed less than a man's height from the floor and directly over the rail. Then, too, tunnels are generally damp or wet, so that a man is rarely well insulated from the ground. As the light at best is poor,

one can not always see the wire as he approaches it, and the space is so restricted that a man walking in the tunnel must keep his head close to the wire when at the same time the most of his attention must needs be given to his footing. Moreover, in climbing into or riding in the cars, most of which in tunnel work are of metal and furnish excellent electrical connection with the rails, one's head must pass close to the live wire. The carrying of metal tools, such as crowbars or drill steel, also picks and shovels with wet wooden handles, is also the cause of many shocks through their accidental contact with the trolley, especially if such tools are carried on the shoulder. It is therefore important, when walking in a tunnel where a trolley wire is installed, constantly to bear its existence in mind and take every precaution to avoid contact with it either by hand, wet clothing, or tools.

In addition to the trolley wire there are in tunnel work other sources from which electrical shocks may be received. Wherever the heading is illuminated by electricity, the lights are usually grouped in a cluster and connected to the main circuit by means of a flexible cable, so that they can be removed easily to prevent breakage during blasting. The wires of the cable are, of course, insulated, but owing to rough usage the insulation is often damaged or scraped off, leaving the bare wire exposed. Even a slight damage of the insulation is often sufficient to permit a considerable leakage of current from which a person handling the cable may receive a severe shock. Such wires are the more dangerous because, supposing them to be protected, one is more apt to handle them carelessly. The men who remove these wires preparatory to blasting and afterwards replace them or otherwise adjust them should examine them closely and not touch any place where the insulation has become damaged.

Shocks are also caused by motors, transformers, or other pieces of electrical equipment that are supposed to be safe but have accidentally become charged, and by switches and other similar devices during adjustment or repair. In handling apparatus of this sort a workman should carefully insulate or otherwise protect himself from the current and should try to handle the apparatus in such a manner that any involuntary muscular reaction from a shock will throw him clear of its live parts, rather than bring him more closely in contact with them. Although electric locomotives are usually in such perfect contact with the rails that a person touching any charged part of the frame will rarely receive a shock, there are times (as, for example, when there is a considerable amount of dirt or sand on the rails) when the locomotive is almost completely insulated from them; in such a case anyone coming in contact with a live part of the frame or of the drawbar, or even with one of the cars coupled to the locomotive, may receive a severe shock, which is apt to be all the more serious be-

cause it is unexpected. For this reason the touching of such equipment should be avoided unless actually necessary.

Mention should be made here of the immediate steps to be taken in case a man has received a severe electric shock and is perhaps lying unconscious and seemingly dead, for it is often possible by prompt treatment to revive and restore a man in this condition who might otherwise fail to recover consciousness. Methods recommended by the bureau are described in Miners' Circular 5.*

FIRE.

The chief danger from fire to the men in a tunnel is the possibility of the buildings at the surface becoming ignited. These structures are, of course, subject to the same causes of fire as ordinary buildings, such as the careless handling of matches or lights, spontaneous combustion of oily waste wherever it is allowed to accumulate, or the short-circuiting of electric wires, not to mention the risk of forest fires in heavily timbered regions. At a large majority of tunnels now being driven the blacksmith shop, the storeroom, the boiler house, and other buildings are situated much closer than the 200 feet that should separate them from the tunnel portal, and in many districts, especially where the winter snowfall is heavy, they are directly connected with the tunnel by snowsheds constructed of wood. At such tunnels, also, means of exit other than the portal are seldom provided, so that in case of fire in these buildings men are penned up in the tunnel and, in the customary absence of a fire door, are in serious danger of suffocation from the gases and smoke produced by the conflagration. It is therefore essential, and in some States it is required by law, that in all tunnels where combustible structures must be erected nearer to the portal than 200 feet there should be a separate exit at least 200 feet away, and that a fireproof door that can be closed from a distance should also be provided. A sufficient water supply should always be maintained to put out a fire, and hydrants with a coiled 1½-inch hose and a nozzle should be placed not less than 40 feet and not more than 100 feet from each building or group of buildings.

Most tunnels, except where timbered, are practically fireproof, and hence underground fires are not common in tunnel work. It is nevertheless important to guard against the dangers of underground fire. Whenever such fires do occur they usually start in some small way, either from candles or lamps being placed too near the posts or caps of a timber set, or from a match or the coals from a pipe thrown into a pile of rubbish, hay or other combustible material that may in turn ignite the timbering. Although such fires can usually be extinguished before any great damage has

* Clark, H. H., Roberts, W. D., Illsley, L. C., and Randolph, H. F., *Electrical accidents in mines; their causes and prevention*, 1911, 10 pp., 3 pls.

resulted, provided their presence is discovered soon enough and there are means at hand to extinguish them, it is much better to prevent the ignition by obviating causes. Therefore, combustible rubbish should not be allowed to accumulate in the tunnel, and any supply of hay for the use of the mules or horses underground should be carefully stored in a shed provided for that purpose, and open lights or smoking should not be permitted in its neighborhood. Candles or torches should never be left burning near timbers, and the practice of wedging a lighted candle between two nails driven into a post should be sufficient cause for the instant dismissal of the guilty person.

WATER.

Water under pressure is another source of danger in tunnel work. Men may be hurt in jumping back to avoid the rocks and other débris often carried with it, or perhaps buried under an accompanying rush of mud and sand. A good example of this may be found in the records of a foreign railway tunnel, where a cleft filled with water, sand, and gravel was encountered and the ensuing sudden and violent inburst filled up more than a mile of the tunnel in a very few minutes, burying 25 workmen beyond all hope of recovery. A somewhat similar occurrence in one of our American tunnels was likewise due to water. The tunnel caved in at a point about 4,000 feet from the heading, but the men working there were warned in time to escape, although they had barely reached safety before the tunnel became entirely closed. When this happened the mass of muck, composed chiefly of soft clay and running shale impervious to water, formed a dam that cut off from the main part of the tunnel the flow of approximately 2,700 gallons per minute. As soon as the part of the tunnel between the cave and the heading became filled with water, the full pressure of the head in the mountain over the tunnel was exerted against the dam, forcing it down the tunnel until the pressure was relieved. The additional length of the débris then offered greater resistance, and it remained stationary until the pressure had again accumulated enough to move it. This process was repeated until 440 feet of tunnel had been filled. Several attempts were made at first to relieve the pressure by inserting a section of ventilating pipe at the top of the dam; but after several men had narrowly escaped being buried by the rush of mud as the dam moved forward this scheme was abandoned and the tunnel was sealed up by a concrete bulkhead, the men being protected by a temporary bulkhead of wood during the construction of the permanent one.

In driving through limestone and dolomite it is not unusual for a tunnel heading to tap immense caves filled with water, mud, and sand. The volume of the fluid mass flowing into the tunnel is deter-

mined by the size of the opening, and its velocity is proportionate to the head. Under the pressure of a head of 300 or 400 feet the cutting action of the rock particles and sand carried by the water soon enlarges even a drill hole to a size that permits the filling up of the heading in an incredibly short time. When a round of shots breaks into a cave of this kind, the heading and perhaps the completed tunnel for a distance of hundreds and even thousands of feet back from the face may be filled so fast that the escape of the workmen is impossible if they are at the face. Fortunately, however, during shot firing, the time of the greatest danger, the men are always out of the heading.

When an underground cave or reservoir filled with water, mud, sand, and loose rock is tapped in a tunnel heading, one of two things occurs; generally the cave or reservoir empties itself completely into the tunnel and, after the flow is over, the solid matter that the flood leaves behind can easily be shoveled up and hauled out. Sometimes, however, the volume of solids is so great that the tunnel is completely choked before the reservoir is emptied. In these cases, when the flow of water ceases, the men are usually set to work cleaning up the material with which the tunnel has been filled, but when this cleaning process advances sufficiently to weaken the dam that holds back the flood a new outburst occurs and, because the passageways have already been opened, the second outbreak is often more violent and dangerous than the first. If this operation were repeated often enough, the cave or reservoir would, of course, be drained and the heading be regained, but in many instances the operation of attempting to regain the heading has been found so dangerous that it has been abandoned and a curved tunnel has been bored to pass around the danger point.

In the dolomite in the Cowenhoven Tunnel, at Aspen, caves of this kind filled with water and dolomite sand were frequently encountered. It was no uncommon thing after a round of shots to have the tunnel completely filled for hundreds of feet back from the face. As soon as the water from the cave that had been tapped drained off, the mud and sand were easily loaded and work in the face was resumed.

An immense cave was tapped by a drill hole in a long crosscut that was being driven from the tunnel to the Della S. mine. The drill hole, under the pressure and cutting action already described, enlarged so rapidly that the men fled from the face, and a few seconds after the opening must have enlarged to a size that permitted the filling of the tunnel with such rapidity that the tunnel cars were hurled back and flattened against the posts. Several unsuccessful attempts were made to regain the face, which finally had to be bulk-headed and the tunnel run around it, as at the Simplon Tunnel in Switzerland.

Numerous caves were encountered in the 1,200-foot level of the Free Silver mine, which was also run through dolomite; but, fortunately, although they must have extended to great heights, their horizontal cross section was very much less than that of the caves 1,200 feet above. When these reservoirs were tapped with a drill hole the water would spout out with such velocity that it was impossible to stay in the face, and in a short time the opening would be worn to a size which sometimes increased the amount of water to be handled by the pumps to 3,000 and even 4,000 gallons per minute. At first the noise from the inrushing volume of water was exceedingly terrifying to the men, but in a short time whenever a cave of this kind was tapped the men simply joined hands to assist each other in maintaining their footing and waded back with the torrent as they would do in crossing an extremely rapid stream. Many narrow escapes occurred, but, due to the precautions taken by the management and workmen, no serious accidents occurred during any of these inrushes.

INTOXICATION.

Although few accidents in tunnel work are traced directly to intoxication of employees, the extent to which it contributes to many mishaps that are attributed to other causes is perhaps too little appreciated. The fact that a man who has put an "enemy into his mouth to steal away his brains" is then much more liable to be careless or negligent of his own safety and the lives of the men around him is so true as to be almost axiomatic. Even a slight amount of intoxication, which might be allowable if the work was to be done on the surface, is dangerous under ground, where it is very apt to be aggravated greatly either by the lack of fresh air or by the heat, neither of which is unusual in tunnel headings. Therefore it is essential that a man in such a condition should not be permitted under ground and, if discovered there, should immediately be sent out of the tunnel by the foreman. Repeated offenses should result automatically in dismissal.

PREVENTION OF ACCIDENTS.

In discussing the prevention of accidents in tunnel work, little is to be gained by arguing whether the manager, the foreman, or the miner is solely to blame for their occurrence. The greater responsibility lying, as ever, with those who have the broader vision, the manager or the superintendent is in duty bound to see that the place where the men are to work shall be made as safe as possible and to insist that they themselves exercise the greatest care and caution in conducting their work. Upon the foreman falls the responsibility of carrying out the manager's orders, of seeing that the men are

instructed in the proper precautions to be taken, and that they are constantly and consistently exercised, and, if necessary, of discharging either temporarily or permanently any man who willfully or habitually disregards them. As for the miner, whose business is shown by statistics to be a hazardous one at best, it is only through the most extreme care on the part of each man, not only for his own welfare but for the safety of his coworkers, that he can hope to escape from the dangers that surround him. Each one, therefore, has his share of the responsibility, and it is only by cooperation between all parties concerned that any progress can be made toward the prevention and reduction of the fatalities and the injuries now encountered in tunnel driving. As it is impossible to reiterate too often the methods of obviating accidents, the following paragraphs are addressed directly to the parties most concerned, in the hope of bringing home to them once more some of the more important preventive measures.

PRECAUTIONS FOR THE MANAGER OR SUPERINTENDENT.

Insist that necessary timbering be done promptly, and always keep an adequate supply of lumber at hand so that no delay may ensue from the lack of it. See that the minimum quantity of explosive is used (in order to prevent unnecessary shattering of roof and walls) and inaugurate a systematic and regular examination of the roof to insure the timbering of loose pieces at once. Have all bent or breaking posts or caps promptly replaced by new ones.

Provide suitable magazines and thaw houses for explosives.*

Do not permit any disregard of the proper precautions as to handling, storing, and using explosives, and see that each man is provided with a copy of such precautions. Do not permit the transportation of detonators or primers to the heading in the same bundle with the remaining supply of explosive for the blast. Have careful tests of the burning rate of the fuse made periodically, especially when a different brand of fuse is purchased, and warn the men of any discovered irregularity. Destroy any damaged fuse at once. Do not store fuse near any source of heat. Prohibit the reloading of a bore hole before it has had time to cool from a previous blast. Give the man who makes the primers the necessary equipment and tools and have him carefully taught how to prepare and waterproof the primers.

Do not purchase caps weaker than 5X for use with gelatin dynamite. See that the proper precautions are taken whenever a missed hole or evidences of one are discovered.

* Specifications for such buildings recommended by the Bureau of Mines are to be found in Technical Paper 18 of the bureau: Hall, Clarence, and Howell, S. P., *Magazines and thaw houses for explosives*, 1912, 34 pp., 1 pl., 5 figs.

Institute a regular and frequent inspection of the valves on the air compressor and insist that any defective valve be promptly and properly repaired even at the cost of a possible shutdown, that there may be no explosion of gas or burning of grease in the receiver or pipe line to produce harmful gases and jeopardize the safety of the men at the heading. Do not delay the installation of adequate auxiliary ventilating equipment when natural accumulations of harmful gases are encountered in the tunnel, particularly when such gases are of an explosive nature. When explosive gases are present, none but safety lamps or their equivalent should be permitted under ground.

Prohibit the men from riding on loaded trips, and, whenever possible, provide special cars for their use, either propelled by hand or drawn by a motor. Do not permit the men to jump on or off moving cars, nor the drivers to "ride the chain." Tell all new men the proper side of the tunnel to take when meeting a trip, and caution them to shield any bright light when so doing.

If there is a trolley wire or other electrical apparatus in the tunnel, caution the men against its danger, and do not allow them to carry tools on their shoulders when passing in or out. See that the cable or wires leading to any temporary or movable cluster of lights in the heading is kept in good repair. Instruct the men, especially the foremen, as to the proper methods of resuscitation in a case of electric shock.

Prohibit the accumulation of combustible rubbish any place in the vicinity of buildings or timbering and see that the supply of hay is properly confined to prevent danger from fire. Unless absolutely necessary, do not construct any wooden buildings nearer than 200 feet to the mouth of the tunnel. If wooden buildings must be built near the mouth, provide a separate exit from the tunnel at least 200 feet away, with a fire door that is arranged to be closed from a distance. In either event, provide an adequate water supply, with hydrants and hoses, at suitable distances from the several buildings.

Exercise great precaution when driving toward a place where a flow of water is likely to be encountered that might carry with it a rush of mud, sand, gravel, or other débris, and take immediate steps for the safety of the men as soon as such a flow is struck.

Prohibit the drinking of intoxicating liquors on property controlled by the tunnel company and institute a system of inspection to prevent any intoxicated man from working in the tunnel, discharging habitual offenders against this rule.

PRECAUTIONS FOR THE FOREMAN.

Insist that the least amount of dynamite required for loading "back" holes shall be used. Do not return to the face after blasting nor per-

mit the men to return without first examining the new roof. Upon arriving at the heading immediately detail as many men as may be required to clean the roof before attempting any other work under it. When passing in or out of the tunnel never fail to inspect the roof, testing any doubtful piece for possible vibration. See that any loose piece of rock is either pulled down at once or properly supported, and never take any chances by postponing the work of timbering no matter how pressing other matters may be, because a few minutes' delay in timbering may cost several lives. Have any timbers showing the effects of too great pressure relieved properly as soon as they begin to fail. When timbering is necessary close to the face, see that the front sets are thoroughly braced and blocked before firing. When the roof "breaks high" fill the space between the lagging and the roof with broken rock or blocking to prevent a large rock from crashing through the lagging upon the men beneath.

See that the men read the precautions to be taken in handling explosives, or have a copy read to them. Do not permit any instance of careless or reckless handling of explosives to go unchallenged and do not fail to discharge men for the first grave offense of this character. Never permit a man to handle dynamite recklessly, either for the purpose of scaring someone or for any other reason. See that the detonators and primers are transported to the heading in boxes separate from the rest of the supply and that they are not placed side by side after arriving. Insist that proper care be used in loading holes and that the tamping be done by pressure rather than by impact. Never allow anything but wooden bars to be used for this purpose. Do not permit a bore hole to be loaded before it has had sufficient time to cool completely from the previous blast.

Warn the men of any change in the rate of burning of fuse. See that they do not mutilate the fuse by rough handling and that they do not crack or break it by placing the primer in the hole fuse end first, or by uncoiling the fuse roughly in cold weather. Do not use fuse that has been stored or kept near a boiler, steam pipe, or other source of heat, or that has been exposed to moisture. See that the fuse is properly coiled close to the hole before blasting, in order that it may not be torn out by the blasts in a near-by hole. Instruct the men as to the proper way to prepare a primer. See that the fuse is cut squarely; that an inch or so of it is discarded; that the grains of powder do not leak out of the end that is inserted into the detonator; that the crimping is done carefully with the proper tool; that the detonator is not buried too deeply in the dynamite; and that caps of sufficient strength are used.

Always count the holes as they are blasted and never fail to inspect the new face for evidences of missed holes. See that any such are detonated properly as soon as they are discovered, even at the

possible cost of some delay. Insist that the shovelers use their picks properly when picking down the muck pile. Keep a close watch for any unexploded dynamite in the muck and have the men do likewise. When such is found remove it carefully to a place of safety and be particularly cautious when a piece of fuse accompanies it. Never start a new hole in the remains of one that has ever held dynamite.

When the presence of any amount of dangerous gases, either from explosives or from natural sources, is suspected see that the men are supplied with fresh air, either by opening the compressed-air line or by breaking into the ventilating pipe, if the current is in the right direction. Do not knowingly remain or permit the men to remain in any atmosphere that will not support a candle flame, because there is no way to determine how bad it may be after the light becomes extinguished. See that the men do not use anything but safety lamps, or their equivalent, in tunnels where explosive gases are encountered, and do not permit any matches or other means of striking an open light to be carried into such a tunnel.

Have the track and roadbed kept in as good condition as possible in order to lessen the risk of derailments. Do not permit men to ride upon loaded trains unless it is absolutely necessary, and in such cases warn them carefully as to the risk being taken. Insist that the men riding in an empty car keep their feet and hands inside of the car and that they watch carefully for low places in the roof. Never fail to discharge any driver caught "riding the chain." See that the men give an approaching train of cars plenty of room, and if animals are used to draw the cars see that the men hide their lights when the animals approach.

Warn the men of the danger from the trolley wire. Familiarize yourself with the proper means of resuscitation after an electrical shock. See that the men are not permitted to carry on their shoulders tools or other instruments that are conductors of electricity. Inspect regularly any cables or wires used for carrying electricity to lights in the heading, or any others that have to be moved frequently, and see that all worn parts are covered with insulating material or replaced if necessary. Do not permit the men to ride on electric locomotives.

See that no piles of combustible rubbish are allowed to accumulate underground, and do not permit the use of candles or torches in the vicinity of hay or other inflammable substances. Do not fail to discharge any men guilty of leaving candles or torches burning near timbers, especially when a candle is wedged between two nails driven into a post.

Exercise special precautions when approaching a place where an inrush of water is to be expected.

Be particularly cautious about drunkenness. Note the men when coming on shift and do not permit a man even slightly intoxicated to go underground. If such a man is discovered in the tunnel send him to the surface at once. Discharge those who are habitual offenders in this respect.

PRECAUTIONS FOR THE MINER.

Do not return to the face of the tunnel without testing the newly exposed roof for loose rocks, and, if any such are discovered, either clean them down yourself or report them to the foreman. Form the habit of carefully examining the roof as you pass in and out of the tunnel, testing doubtful places for vibration; call the foreman's attention at once to any ground that you think should be timbered or to any timbers that need relieving to prevent their breaking.

If you are called upon to use dynamite, do so with great care, observing the precautions outlined in previous paragraphs. Never attempt to scare anyone by reckless handling of explosives and never treat dynamite with roughness. Never place or carry detonators or primers and the rest of the supply of dynamite for the round in the same box or bundle. If it is your duty to assist in the loading of the holes, do this with care, using pressure rather than a blow to tamp the powder in the hole, and be careful never to use too much force in pushing it.

Inquire as to the rate at which the fuse burns, especially when a new brand is being tried, and see that the fuse is cut long enough to give you and your companions time to reach a place of safety. Protect the fuse from mechanical injury, such as scraping, blows, or too great pressure either from falling rocks or from the tamping bar; never use a fuse that has been thus damaged. Never reload a bore hole before it has had time to cool. Do not use fuse that you know has been stored near a boiler, steam pipes, or other source of heat or one that has been exposed to moisture. If you prepare the primer, see that an inch or so is cut squarely from the end of the fuse before it is put into the detonator; that no powder runs out of the end of the fuse during this process, and that the detonator is properly crimped around the fuse. Under no circumstances use anything but the regular crimping tool for this purpose.

Always inspect each new face for evidences of a misfire, and, if one is discovered, call the foreman's attention to it immediately, so that he may have it detonated. Never attempt to pick out the material in such a hole; either explode it with a primer or, if this can not be done, drill and fire another hole at least 2 feet away. Use great care in removing any unexploded dynamite from the muck pile, and be especially cautious if a piece of fuse is discovered near

it, for this may mean that there still is a detonator in the cartridge. Never handle a pick like a sledge hammer; pull or scrape the material down rather than strike it with the pick. Do not start a new hole in the remnants of a former one that has ever held dynamite, for there is always a chance that the dynamite may not have been detonated.

Whenever you feel that you are inhaling fumes from dynamite that has burned, or any other harmful gases, try to get to fresh air as soon as possible; the quickest way to do this is often to open the compressed-air line, or to break the ventilating pipe, if you know that the current is in the right direction. Never use anything but a safety lamp or a portable electric lamp in a tunnel where explosive gases are known to exist, and do not carry any other means of striking a light into such a tunnel.

Never attempt to ride upon a full car or a loaded trip; and when riding in empty ones see that your feet and hands are well inside and that your head is low enough to clear the roof at all places. Learn which side of the tunnel has the most room, and when a trip of cars approaches allow yourself as much clearance as possible. If the trip is drawn by an animal, hide any bright light you may be carrying. If it is your duty to drive a horse or mule or to run a locomotive, try to do everything possible to prevent derailments; report any places where the track or roadbed is in bad condition. Remember that the front end of the trip is the most dangerous place you can stand, so that if this is necessary, you must take extra care; never under any circumstances ride with one foot on the chain by which the cars are being pulled. Take care that the animal does not step on you or kick you, and speak to him before approaching him from the rear. In placing a derailed loaded car back upon the rails, take care not to strain or otherwise injure yourself in so doing; keep your feet and hands in a safe position, and see that the car does not topple over and crush you against the sides of the tunnel.

Bear constantly in mind that the trolley wire is dangerous and that you must pass within a few inches of it when going in and out of the tunnel, often when your attention must be given to your footing. This danger should be especially avoided when climbing into cars. When you are in a tunnel where there is a trolley wire never carry tools, drill steel, or anything else that is metal or wet on your shoulders. Do not handle any electrical equipment unnecessarily nor ride on electric locomotives. Never cause anyone to receive an electric shock; it is never possible to foretell its results. If it is your duty to repair electrical apparatus, see that you are properly insulated, or that the current is cut off and can not be turned on without your knowledge; keep your hands and body in

such a position that a recoil from an accidental shock will throw you clear of any charged part of the apparatus. In removing and replacing the temporary cluster of electric lights in the heading be careful not to touch any bare or injured place in the wires, and call the foreman's attention to any damaged place you may discover. Familiarize yourself with the methods of reviving a person injured by electric shock and put them into practice as soon as possible whenever necessity occurs.

Do not smoke or throw a lighted match near any pile of inflammable rubbish either in a building or near timbering, and do not carry a candle or a torch near any piles of hay. Never wedge a candle between two nails on a post or other piece of timber; many disastrous mine fires have started in just this way.

Never take a drink of liquor before or during working hours, and do not hesitate to report any man you see doing so or who is in an intoxicated condition; your safety and perhaps your life may be sacrificed to his carelessness when under the influence of liquor.

SURFACE EQUIPMENT FOR DRIVING TUNNELS.

The equipment on the surface at tunnels that are being driven usually centers around a power house. Ventilating machinery for supplying fresh air to the men working in the tunnel is usually placed here, with the blacksmith and repair shops near by. Other surface equipment consists of storehouses and buildings in which the tunnel crew are housed and fed.

POWER EQUIPMENT.

SOURCES OF POWER.

Although the power for driving a tunnel may be obtained from various sources, in general practice at present it is produced primarily from either steam or flowing water. Although, as far as could be ascertained, the gas producer used in connection with internal-combustion engines has been installed at only one tunnel, nevertheless it offers a possibility as a source of power that will have to be seriously considered in the design of future plants. Within the last few years, as its advantages have become better appreciated, the gas producer has developed rapidly—so rapidly, in fact, that few people realize that it is to-day as reliable and substantial a piece of apparatus as an ordinary boiler, that its consumption of fuel is only one-third as great, and that the labor to operate it need not be one whit more skilled. The gasoline engines occasionally used to furnish power in tunnel operations have been confined either to temporary plants or to small and isolated units of machinery. In localities where petroleum is cheap it is probable that an oil engine of the Diesel type, with its wonderful fuel economy, may be found the cheapest means of producing power. Electricity, especially where it is used at tunnel plants to operate prime moving machinery, is sometimes considered a source of power; but as the current so employed has to be generated elsewhere, usually from steam or water, or possibly from producer gas, petroleum, or gasoline, electricity is merely a convenient form of power for transmission, not a source of power.

PRODUCTION OF POWER.

WATER POWER.

In tunneling, the machines most frequently employed for the utilization of water power are of the impulse type. Such a wheel is driven by the force of a stream of water issuing from a nozzle,

acting against vanes or buckets on the circumference of the wheel, and is well adapted for use with a relatively small volume of water under high head. The efficiency of the machine is dependent upon the way the vanes or buckets reverse the direction of the water discharged upon them; hence they usually conform to a curve which is very carefully designed to avoid loss of power through eddies and friction as the water strikes the vane. There may be more than one nozzle in order to obtain greater power or, if high rotative speed is desired, a small wheel with multiple nozzles may be substituted for a large one. In order to obtain the best results the peripheral speed of the cups or vanes should be between 42 and 48 per cent of the speed of the water issuing from the nozzle. Impulse wheels are manufactured in many different designs, sizes, and speeds, adapted for working under widely diverse conditions. Those observed at different tunnels examined for this report were, as far as could be learned, giving very satisfactory service.

The turbine wheel, in which the force of the water is made to act through suitable guides upon all the vanes or blades simultaneously, affords another means of utilizing water power and may be designed for either high or low heads. Its use is limited, however, especially with high heads, to localities where clear water is available, because of the abrasive action of sand and grit upon the guides. With low heads this action is not so marked. As the water power available in the vicinity of tunnels and adits is in most cases from streams furnishing high heads and at certain seasons of the year carrying large amounts of sediment, the use of turbine wheels for such plants is not feasible unless large settling basins can be provided.

The hydraulic compressor, converting the energy of water directly into compressed air, offers a third method for utilizing water power. The earliest type operates on the principle of the hydraulic ram, in which a column of water is allowed to acquire velocity and is then suddenly checked, developing intermittently for a short space of time pressure much greater than that due to the head of the column of water. This pressure is employed in compressing air. Sommeiler, in about 1860, designed a machine of this type for use at the Mount Ceniz Tunnel. Such compressors require rather high heads and have low efficiency. Although conditions might be such as to make the use of compressors of this type desirable, the water power they require can generally be utilized more advantageously in some other manner.

The hydraulic compressor recently developed by C. H. Taylor,* introducing air into a column of water and compressing it as the air and water fall together to the bottom of a shaft where the air is

* See article on Taylor hydraulic air compressor, *Compressed Air Mag.*, June, 1910, p. 5675.

the square inch and is transmitted by a 20-inch main to the mines 9 miles distant. The capacity of the plant is the compression of 40,000 cubic feet of free air per minute to 120 pounds per square inch.

The familiar overshot, breast, and undershot wheels are not used to drive machinery for tunnel work because of their large size in proportion to the power developed and because of the trouble of their maintenance.

The following table, which is based upon actual results, shows the efficiency of different types of water motors:

Percentage of theoretical horsepower realized by various water motors.

	Per cent.
Impulse wheels.....	70 to 85
Turbine wheels.....	75 to 85
Overshot wheels.....	60 to 65
Breast wheels.....	50 to 60
Undershot wheels.....	30 to 50

Results obtained at the testing flume of the Holyoke (Mass.) Water Co., whose tests are taken as standard by American engineers, show efficiencies for turbine wheels under favorable conditions of over 80 per cent,^a but this is unusual, the figures above being much nearer the efficiency obtained in ordinary practice. The efficiency of hydraulic compressors of the ram type is about 30 to 40 per cent, whereas the Taylor compressor at Cobalt is said to utilize at least 75 per cent of the theoretical power of the water.

STEAM ENGINES.

Steam engines are of two types, reciprocating and turbine. In the reciprocating engine power is developed by the pressure and expansion of steam in a cylinder acting against a moving piston. Such engines may be either simple or compound, both forms being used in tunnel plants. In simple reciprocating engines the total expansion of the steam and consequent reduction of pressure takes place in one cylinder, whereas in the compound type a part of the expansion takes place in the first cylinder and the steam, under somewhat reduced pressure, is expanded further in a second cylinder, necessarily larger because of the lower pressure of the steam.

The steam turbine is similar in principle to the water wheel, except that steam instead of water is the motive fluid. Among its advantages are economy, small size per unit of power, and freedom from vibration, and its use is steadily increasing on both land and sea. Modern steam turbines in sizes of 250 to 500 horsepower, with a steam pressure of 150 pounds and a 28-inch vacuum, will develop

^a Larner, C. W., Characteristics of modern hydraulic turbines: Trans. Am. Soc. Civ. Eng., vol. 66, March, 1910, p. 322.

a kilowatt-hour with a consumption of 18 to 20 pounds of steam. A recent series of shop tests* on a 300-kilowatt Swiss condensing turbine showed that with 112½ pound of steam and a 96.6 per cent vacuum a kilowatt-hour was produced with 16.1 pounds of steam. The difficulty of reducing the high rotative speed of the turbine engine down to the restricted speed of reciprocating machinery has prevented until recently the use of turbine engines in tunnel installations, but with the advent of the turbocompressor we may expect to see them dividing the field, or, perhaps, entirely displacing the cumbersome reciprocating plants now in vogue.

INTERNAL-COMBUSTION ENGINES.

Internal-combustion engines develop power from the pressure produced by the explosion or rapid combustion (confined in a suitable cylinder) of a mixture containing the proper proportions of air and a gasified fuel. The source of the fuel gas may be gasoline, kerosene distillate, or even crude petroleum, or the gas may be generated from coal by distillation in a retort or by a gas producer. The engines are usually designated by the kind of fuel for which they are adapted, as, for example, oil engines, gasoline engines, or producer-gas engines. As far as could be ascertained, the latter two are the only types now used in tunneling.

Although the gasoline engine has been developed with wonderful rapidity during the last 25 years in connection with the automobile industry, the use of engines of this type for tunnel work has been confined to a very limited field, namely, the operation of machinery in isolated or not easily accessible localities. As prime movers for tunnel plants of any magnitude they can not compete, under most circumstances, with machines using other sources of power, and on this account their application has been confined either to enterprises small in scope or to the temporary and early development stage of larger projects, for which they are sometimes installed to begin the work at places by nature inaccessible for nonportable units of heavier machinery, pending the construction of a special roadway or a transmission line. Most manufacturers of air compressors have recently begun to supply air compressors directly driven by internal-combustion engines, although as yet only the smaller sizes of gasoline engines are being used. With suitable adaptations the principle might be applied equally well to the larger sizes using oil or gas as fuel. Within the past three years gasoline locomotives have been successfully employed in coal mines, and, according to the manufacturers, there are now over 100 of them in operation. These machines are equally

* Described in a letter from the manufacturer to the authors.

suitable for use in tunnel work and will, no doubt, be employed extensively for this purpose in the near future.

The only use of producer-gas engines in tunnel work to date, so far as learned, was in England at the power plant for the tunnel recently completed under the Thames River, connecting North and South Woolwich. As the tunnel was driven by the use of air locks absolute reliability in the power plant was required to avoid a decrease of pressure that might result in serious damage to the tunnel. At this plant, as described in "The Engineer,"^a three engines, each of 150 brake horsepower, when running at 180 revolutions per minute, were supplied with gas from suction producers using Scotch anthracite coal and were connected to a central shaft which transmitted power to 4 air compressors and 4 dynamos. The plant was operated continuously from July, 1910, until the end of December, 1911, except during October and November, 1910, when after the vertical shaft had been completed the plant was purposely stopped while preparations were being made to start tunneling.

The gas producer is discussed in detail in Bureau of Mines Bulletin 13.^b The reader is also referred to the bibliography accompanying this report and to the bulletins noted below, which are issued by the Bureau of Mines:

Bulletin 4. Features of producer-gas power-plant development in Europe, by R. H. Fernald. 1910. 27 pp., 4 pls., 7 figs.

Bulletin 7. Essential factors in the formation of producer gas, by J. K. Clement, L. H. Adams, and C. N. Haskins. 1911. 58 pp., 1 pl., 16 figs.

Bulletin 16. The uses of peat for fuel and other purposes, by C. A. Davis. 1911. 214 pp., 1 pl., 1 fig.

Bulletin 31. Incidental problems in gas-producer tests, by R. H. Fernald, C. D. Smith, J. K. Clement, and H. A. Grine. 29 pp., 8 figs. Reprint of United States Geological Survey Bulletin 393.

Bulletin 55. The commercial trend of the producer-gas power plant in the United States, by R. H. Fernald. 1913. 93 pp., 1 pl., 4 figs.

Although, as far as could be learned, oil engines of the Diesel type have not been employed in tunnel power plants their marked success in other fields warrants the discussion of the possibility of their use in tunnel work. The Diesel differs from other internal-combustion engines in that instead of drawing into the cylinder an explosive mixture of air and a combustible gas (such as producer gas, gasoline vapor, kerosene, or even crude petroleum previously volatilized by heat) and then compressing this mixture and exploding it by means of an electric spark, or some other suitable device, the Diesel engine compresses air alone, and when this is under high pressure (approx-

^a The Engineer (London), Temporary power plant for Woolwich footway tunnel: Jan. 12, 1912, pp. 46-47.

^b Fernald, R. H., and Smith, C. D., Résumé of producer-gas investigations, Oct. 1, 1904, to June 30, 1910. 1911. 398 pp., 12 pls., 250 figs.

mately 500 pounds per square inch, which is much greater than that usually attained in other types of internal-combustion engines) injects into the cylinder a spray of finely atomized oil. During the compression of the air to the required pressure it reaches a temperature of more than 1,000° F., more than sufficient to ignite the oil instantly without the use of an electric spark, hot plate, or other similar device.

The chief advantage of the Diesel engine is economy of fuel. It is a well-known fact that the rapidity and completeness of combustion are increased by pressure; it is not surprising therefore that under the high pressures in this machine excellent results can be obtained from a small amount of oil. The extremely fine atomization of the fuel, due to its being injected by compressed air under a pressure of 300 to 500 pounds per square inch higher than that in the cylinder, is undoubtedly another important factor. And again, as scavenging is effected by air only instead of by a mixture of air and fuel, as is the case in other types of internal-combustion engines, there is no possibility of fuel being lost through the exhaust valves during this process, a saving that is extremely important in engines designed for a two-stroke cycle.

In addition to the advantage of fuel economy, however, the Diesel engine does not require frequent cleaning, as is the case with oil engines depending upon a hot plate or a similar device for the ignition of the explosive mixture. It also dispenses with the carbureter, so necessary for the gasoline engine and always a source of trouble and annoyance. In addition, as the mixture being compressed in the cylinder of a Diesel engine is not explosive, allowance does not have to be made in the design of the cylinder and other parts for undue stresses and strains that might result from a premature ignition of the charge, of not infrequent occurrence in other engines. Although some provision can, of course, be made for these shocks, their force and violence can not always be foreseen and there have been many instances of disastrous results arising from them.

The principal disadvantage of the Diesel engine, on the other hand, is that of high first cost, and this would prohibit its use for tunnel work of short duration. However, the price has recently been greatly reduced abroad and it is certain to be reduced in America, now that manufacturers in this country are equipped with suitable apparatus and prepared to execute the high class of workmanship required in its construction; hence in the near future this drawback may disappear. But even now, if the time required for the completion of the work is to be long enough or the amount of power to be used is great enough to warrant a heavy initial outlay in order to effect a saving in operating cost, the choice of a Diesel engine should be seriously considered.

ELECTRIC MOTORS.

Electric motors may be designed either for direct or alternating current. Where used as prime movers at the tunnel plants visited they were of the alternating-current type only and operated at comparatively low voltages, usually 440 volts. Their power was generally transmitted to the machinery by means of belts, but at one or two places on the New York Aqueduct direct-connected electric-driven air compressors were noticed.

TRANSMISSION OF POWER.

Electricity, because it can be economically transmitted long distances, is the favored means of transmitting to a tunnel the power generated at some remote station. It possesses one well-known disadvantage, that of occasional interruption, especially where distances are great and tension high, because of unavoidable hindrance to service through storms. On the other hand, producer gas has possibilities that deserve serious consideration. Its transmission has been recently taken from the realm of mere conjecture and demonstrated as practical in the Pittsburgh (Pa.) district where natural gas is piped for distances of 200 miles, and in England, where producer gas is supplied to points within a radius of 160 miles with transmission losses even less than those of electricity. In its application to tunnel operations, producer gas can be generated in a plant conveniently situated on a railroad siding or some other readily accessible place, and the power piped to internal-combustion engines at the tunnel portal.

Where distances are comparatively short (that is, less than 5 miles) electricity is rivalled by compressed air, and the competition grows more keen as the length of the transmission system decreases. This is possible because compressed air, in spite of its low efficiency and high cost, is necessary where pneumatic drills are employed; and even were electric transmission chosen, air would have to be compressed by electricity at the tunnel. Hence it is the usual practice to produce the compressed air at once, thus avoiding the extra machinery and the additional operating losses of electric transmission.

CONSIDERATIONS INFLUENCING CHOICE OF POWER.

A number of factors enter into the choice of power for tunneling operations. To begin with, the plant is usually short lived. Then, too, such local conditions as accessibility, distance from a railroad, the availability of water power, etc., must be considered. Each method of deriving power has also certain peculiarities that render it particularly adaptable to some conditions. Among these may be men-

tioned the cost of installation, of labor, of fuel, of interest and depreciation, and other operating expenses. Aside from all this, it is often necessary to decide between the production of power at the plant or elsewhere and the purchase of power from an established hydroelectric company. Some of these factors are discussed briefly below.

DURATION OF PLANT.

At many tunnel power plants, in direct contrast with those used in manufacturing, the equipment is required only for the comparatively short time of actual tunnel construction. Thus it becomes a delicate problem to determine just how far one is justified in the purchase of machinery and apparatus for utilizing all the various economies that may be effected in the production of power. It is difficult to decide whether it would not be better in the end to install less costly machinery that would necessitate slightly higher expense in operating and maintaining than to tie up extra capital in equipment that would be of no further use when the tunnel is completed. Of course, the shorter the probable life of the plant the more would one be justified in such a course. Although, if the equipment can be transferred upon the completion of the tunnel to other projects, this would so prolong its period of usefulness that the original expenditure of capital could properly and with true economy be greater. A notable instance of this was observed on the Los Angeles Aqueduct, where, as far as possible, upon the completion of one of the numerous tunnels, the equipment was transferred and used in power plants at other tunnels whose construction had not yet begun. If a central station for generating a large amount of power is being considered, the purchase of apparatus with regard to ultimate economy in operation is again the far-sighted policy. This was the case at the Rondout Siphon Tunnel, but at the average mining tunnel the converse is more often likely to be true.

ACCESSIBILITY OF TUNNEL.

Many tunnels are driven at places very difficult of access. They may be so situated as to make the installation of heavy machinery no easy matter, as, for example, where the road from the nearest railroad is poor and has steep grades; or they may be at a great distance from the nearest siding, so that if the power plant requires coal, the delivery of this fuel is not only costly, but also uncertain and difficult in some seasons of the year. Such conditions are favorable for the adoption of power transmission in some form from a waterfall or rapid, if there is one near enough, or, lacking these natural advantages, from a fuel plant installed at some point more readily accessible.

COST OF INSTALLATION.

The cost of installing water wheels depends entirely on local conditions, which are never twice alike. Where high heads are available and the quantity of water required is not large, water can be conveyed to the water wheel by small flumes or pipes, which are comparatively inexpensive. For example, at the Carter Tunnel (Pl. I, *A*), with an available head of 145 feet, a flume 16 by 48 inches inside and 5,000 feet in length is sufficient to supply the 200 horsepower developed. At the Laramie-Poudre Tunnel, with a static head of 268 feet, 400 horsepower is conveyed to the tunnel plant by a wooden-stave 22-inch pipe line, 8,500 feet in length. The Utah Metals Tunnel obtains water from two sources: The first gives a 700-foot head, requires 2,500 feet of 12-inch, 1,900 feet of 10-inch, and 100 feet of 8-inch spiral, riveted steel pipe, and furnishes 170 horsepower; the second, giving a 750-foot head, requires 2,000 feet of 12-inch, 1,000 feet of 8-inch, and 3,000 feet of 6-inch pipe to produce 55 horsepower.

Where heads are low retaining dams are usually necessary. At best these are expensive, and their cost increases enormously with their height. With low heads larger flumes are required to convey the greater quantity of water. At the Nisqually Tunnel (Pl. I, *B*), a low dam and a 6 by 8 foot wooden flume 1,200 feet long are used. The water is delivered under an effective head of 29 feet to a turbine wheel that generates 1,000 horsepower. One has only to consider some of the very expensive dams on the larger rivers, furnishing power for manufacturing purposes, in order to realize how great the cost of installation may be where low heads are utilized. It is fortunately true, however, that where water power is obtainable for tunnel work the heads are usually high and the less expensive flumes or pipe lines of moderate length can be utilized.

The cost of the machinery actually within a tunnel power house is greater for steam than for water power or electricity; but if, as should be done to make the figures truly comparable, the cost of the dam and the flume (or of the transmission line for electricity) be taken into consideration, the advantage is usually reversed. It is somewhat cheaper to install a steam plant than one using producer gas and having engines of the same capacity, but the difference is not great. Fernald,^a after a study of many tables of costs, applying to other uses, however, than tunnel work, concludes that "complete producer-gas installations for the larger plants, say, from 4,000 to 5,000 horsepower, cost about the same as those of first-class steam plants of the same rating. With smaller installations the balance is probably in favor of the steam plant." As it is not customary in

^a Fernald, R. H., The commercial trend of the producer-gas power plant in the United States: Bull. 55, Bureau of Mines, 1913, p. 26.

tunnel work to install machinery designed to effect all the refinements of steam economy found in permanent plants, it is probable that the first cost of the average steam plant for tunnel work is less than those upon which Fernald's estimates are based, in which case the comparison would be even more favorable to steam. This is partly offset by the fact that the price of gas producers and engines is constantly being lowered, and by the fact that the cost of actually placing the machinery would be less for the gas producer.

The initial expense of installing any of the various systems for transmitting power is dependent upon two factors: The cost of the machinery required to produce the power, to convert it into the form suitable for transmission, and to reconvert it into the form adapted to the machines using it; and the cost of the transmission line. Except for a slight increase in capacity, to provide for losses in transmission, the factor of machinery cost under any given conditions is independent of the distance over which power is to be delivered, but the cost of the line increases somewhat faster than its length, other things being equal. If the power is required ultimately for the operation of air drills, practically the same size of compressor will be necessary whether electric, air, or gas transmission is employed, and the cost of boiler, engine, and foundations, in the case of electricity or air, will approximately balance the cost of the producer, engine, and foundations for gas. Air transmission requires practically no other machinery than that just mentioned, but gas, on the other hand, needs a blower of some sort to force it through the line, and electricity requires, in addition, a generator, motor, transformers, and the extra foundations. A list of the three forms of power transmission arranged in the order of increasing machinery cost would be air, gas, and electricity.

The cost of an electric transmission line may be divided into three parts: First, the conductor and connections; second, insulation; and third, erection of the line. Although a detailed discussion of this subject is beyond the scope of this bulletin, it may be stated that, for a given power loss and a given distance, the weight of the conductor required to transmit a definite amount of power is inversely proportional to the square of the voltage employed. On the other hand, the cost of insulation increases rapidly with the potential, and the cost of erection, complicated by steel towers and special precautions, is greatly augmented at high voltages. Thus the economical transmission of a given amount of power for a stated distance is limited by the maximum voltage that may be used without the increased cost of installation and erection destroying the saving in the cost of copper.

Relative to the most advantageous voltage for the short distances and small amounts of power commonly employed in tunneling opera-



A. END OF FLUME, MILL, DUMP, AND OTHER SURFACE FEATURES AT THE CARTER TUNNEL.



B. POWER HOUSE, PORTAL, AND PART OF FLUME, NISQUALLY TUNNEL, ALDER, WASH.

tions, the following rule of thumb will give results closely approximating those of the most careful calculations: Multiply the distance to be traversed in miles by 1,000 and select the voltage of the commercial size of transformer nearest to this figure. The standard voltages of transformers now in use are 220, 440, 660, 1,100, 2,200, 6,600, 11,000, 22,000, 33,000, and 66,000. If the distance from the power station to the tunnel plant is 5 miles, select a voltage of 6,600; if the distance is 10 miles, a voltage of 11,000. If the distance falls midway between transformers steps, use the voltage that will permit most ready sale of the apparatus when the work is completed.

As there are certain difficulties in the construction of direct-current generators for voltages higher than 600, alternating current is generally employed for transmission lines. An important additional advantage of alternating current is the ease with which the potential may be changed from low to high, or vice versa. Where electricity is transmitted at high tension it is usually generated at a comparatively low voltage, "stepped up" by transformers to the desired potential for the line, and "stepped down" by transformers at the tunnel plant.

The following figures, which show the installation cost of an electric-transmission line for different voltages and distances, assuming approximately 10 per cent drop in the line, are based upon data kindly furnished by the General Electric Co.:

Cost of installation for electric-transmission line for different voltages and distances.

1. 200 horsepower; 1 mile; 440-volt, direct current:	
Poles, cross arms, insulators, and fittings (poles spaced 100 feet)---	\$375
33,000 pounds copper cable, 500,000 circular mils (4 conductors required), at 18½ cents per pound-----	6,025
Cost of erection-----	300
Total-----	6,700
2. 200 horsepower; 1 mile; 440-volt, 3-phase, 60-cycle, alternating current:	
Poles, cross arms, insulators, and fittings (poles spaced 100 feet)----	415
34,000 pounds copper cable, 350,000 circular mils (6 conductors required), at 17½ cents per pound-----	6,035
Cost of erection-----	375
Total-----	6,825
3. 200 horsepower; 1 mile; 1,100-volt, 3-phase, 60-cycle, alternating current:	
Poles, cross arms, insulators, and fittings (poles spaced 125 feet)---	385
5,100 pounds copper cable, B. & S. No. 0 (3 conductors required), at 17½ cents per pound-----	905
Cost of erection-----	285
6 transformers, 1,100 to 440 volts, with switches, etc., erected-----	2,900
Total-----	4,455

4. 200 horsepower; 5 miles; 1,100-volt, 3-phase, 60-cycle, alternating current:

Poles, cross arms, insulators, and fittings (poles spaced 125 feet)---	\$1,870
122,000 pounds copper wire, B. & S. No. 000 (9 conductors required), at 17½ cents per pound-----	21,650
Cost of erection-----	1,580
6 transformers, 1,100 to 440 volts, with switches, etc., erected-----	2,900

Total----- 28,000

5. 200 horsepower; 5 miles; 6,600-volt, 3-phase, 60-cycle, alternating current:

Poles, cross arms, insulators, and fittings (poles spaced 125 feet)---	1,870
6,500 pounds copper wire, B. & S. No. 6 (3 conductors required), at 17½ cents per pound-----	1,150
Cost of erection-----	1,080
6 transformers, 6,600 to 440 volts, with switches, etc., erected-----	3,700

Total----- 7,800

6. 200 horsepower; 25 miles; 6,600-volt, 3-phase, 60-cycle, alternating current:

Poles, cross arms, insulators, and fittings (poles spaced 125 feet)---	9,350
103,000 pounds copper wire, B. & S. No. 1 (3 conductors required), at 17½ cents per pound-----	18,300
Cost of erection-----	5,150
6 transformers, 6,600 to 440 volts, with switches, etc., erected-----	3,700

Total----- 36,500

7. 200 horsepower; 25 miles; 22,000-volt, 3-phase, 60-cycle, alternating current:

Poles, cross arms, insulators, and fittings (poles spaced 125 feet)---	9,900
33,000 pounds copper wire, B. & S. No. 6 (3 conductors required), at 17½ cents per pound-----	5,860
Cost of erection-----	5,190
6 transformers, 22,000 to 440 volts, with switches, etc., erected-----	5,200

Total----- 26,150

Freight, right of way, surveying, and engineering costs are not included in the above data.

The Pneumo-Electric Machine Co.* has estimated that if compressed air were used to transmit 200 horsepower 1 mile, 10 per cent loss at 80 pounds pressure being allowed, an 8-inch pipe would be required, which, at \$1.78 per foot, would cost \$8,400. Calculations show that in order to transmit the same amount of power in the form of producer gas containing 120 British thermal units per cubic foot, the required pipe would have to be only 4 inches in diameter, costing, at 70 cents per foot, approximately \$3,700. To both these values should be added the expense for laying the line, but this figure would be small compared to the cost of the pipe.

* Cost of power transmission; electricity versus compressed air; Min. and Sci. Press, May 14, 1910, p. 700.

Where the power is ultimately required for use in air drills and power is to be transmitted only for short distances compressed air is the cheapest of the three methods as regards installation cost, the higher cost for the machinery required by the other systems more than balancing the expensive air-pipe line. Producer-gas transmission, with its cost for machinery slightly greater than that for air, yet less than that for electricity, and its line factor just the reverse, is best suited for the medium distances—beyond the economical range for air, but still too short, on account of extra machinery, for electricity. For long distances, on the other hand, electric transmission at high tension is, of course, preeminent.

LABOR REQUIREMENTS.

Tunnel power plants are generally not large enough to require the entire time of even one operator; hence it is impossible to prevent their being overmanned. Consequently the amount of labor required does not as a rule seriously affect the choice of power. At a tunnel plant using water wheels, hydraulic air compressors, or electric motors as prime movers one man per shift is sufficient. Even then, as was the case at the Laramie-Poudre Tunnel, it is not unusual to make the shifts 12 hours long, thus requiring only two men per day; or, as at the Carter Tunnel, to assign the engineer other work for a part of his time. If the results obtained from its use in other work be accepted, a producer-gas plant would require no more exacting attention, one man per shift at plants that develop as high as 750 or 1,000 horsepower being required. A steam plant of the same capacity, on the other hand, would require at least two firemen in addition to the engineer. In larger steam installations the labor required per horsepower generated is naturally not so great. For example, at the Rondout siphon tunnel 8 men per 8-hour shift were able to attend to a steam plant rated at 4,000 horsepower and containing 10 air compressors, each having a capacity of 2,400 cubic feet.

FUEL CONSUMPTION.

If the charge for delivering fuel be included the cost of fuel at most tunnel plants is high; hence the quantity required is of great importance. Steam plants require much more coal than gas plants of the same size, for although large steam plants, with every means for effecting thermal economies, may be operated with as little as 2 pounds of high-grade fuel per brake horsepower-hour, in small plants such as are used in tunnel work a fuel consumption as low as 3 pounds would be exceptional, and 4 and 5 pounds is more probable. With producer gas, on the other hand, it has been repeatedly demonstrated that internal-combustion engines can be operated on less than 1 pound of coal per brake horsepower-hour, and at the best

plants this figure runs as low as three-fourths of a pound. The consumption at the Woolwich Tunnel plant during a test was 0.727 pound of Polmaise Scotch anthracite per brake horsepower-hour. The small internal-combustion engine has the additional noteworthy characteristic of being decidedly efficient in small sizes. A gas engine of 50 to 60 brake horsepower has little greater fuel consumption per horsepower than a large gas engine of 500 or 1,000 brake horsepower.

THERMAL EFFICIENCY OF ENGINE.

The comparatively high fuel consumption of the steam engine is due to its low thermal efficiency. Although large and economically operated steam plants may realize 12 to 15 per cent of the energy of the coal, 5 per cent is much nearer the value generally obtained in tunnel work. The following table shows the distribution of the average heat losses for one year at a well-conducted steam plant where the thermal efficiency at the flywheel was 10 per cent:

Loss of theoretical heat energy at a steam plant.

	Per cent.
Losses due to imperfect combustion, heat absorbed in ashes, moisture, etc., heat in flue gases, radiation, etc.-----	25
Loss due to latent heat in exhaust steam-----	60
Loss in steam pipes and auxiliaries-----	3
Loss due to friction in steam engines-----	2
	90

The producer-gas engine, on the other hand, has a much higher efficiency, 20 to 30 per cent being not unusual in actual practice. Recent exhaustive shop tests of a number of first-class foreign-built producer-gas engines ranging in power from 70 to 120 horsepower gave thermal efficiencies at full load of 31.3 to 34.9 per cent and a coal consumption of 0.72 to 0.623 pound per brake horsepower-hour. The following table shows the distribution of losses in a producer-gas plant operating with similar economy to the steam plant cited above:

Thermal losses in producer-gas plant.

	Per cent.
Loss in gas producer-----	15
Loss in water jacket-----	21
Loss from radiation and friction-----	4
Loss in exhaust gases-----	35
	75

COST OF ELECTRIC CURRENT.

If the line of an established electric-power company runs near enough to the tunnel plant, it may be cheaper to buy power than to

generate it. Such a concern is in a position to utilize a waterfall too distant to warrant its development for a single-tunnel project, and by distributing a large amount of power among many permanent customers is enabled to sell it very cheaply. The price of current usually ranges from $1\frac{1}{2}$ to 2 cents per kilowatt-hour. On the Los Angeles Aqueduct the power used at all the tunnel plants was obtained from a private transmission line operated by a separate department of the aqueduct organization, and a flat rate of $1\frac{1}{10}$ cents per kilowatt-hour for power was charged against each tunnel, which was estimated to be sufficient to operate the system and eventually pay for its installation. At a tunnel in Colorado a flat rate of \$2.50 per horsepower-month is charged, to which is added $1\frac{3}{8}$ cents per kilowatt-hour used. On a 24-hour day basis this is equivalent to $1\frac{1}{2}$ cents per kilowatt-hour. At another tunnel in Colorado, 2 cents per kilowatt-hour is the price of current. At a third, the power for the compressor costs \$5.50 per horsepower-month, which is equivalent to 1 cent per kilowatt-hour on a 24-hour day basis; but at the same tunnel 2 cents per kilowatt-hour is charged for the current used in the motor generator set that operates the trolley system, making the average cost for the total power used approximately $1\frac{1}{2}$ cents per kilowatt-hour. At one tunnel plant using much power the current is said to have cost only $\frac{1}{3}$ cent per kilowatt-hour, an exceptionally low figure, but in this case other considerations were involved that really made the cost of the electricity greater.

The following schedule is used by a number of western hydroelectric companies, who claim that this rate and method of making a charge is "fair and rational."

Schedule of charges for electricity.

Horsepower.	Fixed charge per month per horsepower of maximum demand.	Energy charge.
For the first 100.....	\$3.25	Add for all energy used as shown by meter 13 mills per kilowatt-hour for the first 40,000 kilowatt-hours used each month, and 5 mills per kilowatt-hour for all additional energy.
For the next 400.....	2.25	
For the next 500.....	1.75	
For all additional.....	1.00	

The maximum demand is determined by the company by meters, disregarding starting peaks and peaks due to short circuits or accidents to user's apparatus.

COST DUE TO INTEREST AND DEPRECIATION.

The charge for interest per unit of power is dependent upon the amount of capital invested, but that for depreciation depends on several factors. In the case of water power a dam or a ditch would

have little salvage value after the completion of the tunnel; more might be realized from a pipe line or flume, and still more from the machinery in the power house. A similar analysis may be made for other means of producing power. Both interest and depreciation charges are dependent also on the number of hours the plant is used daily, it being evident that if a plant be used 24 hours instead of 12 the same total cost for interest and practically the same total loss by depreciation will be distributed over double the number of horsepower-hours, and hence be proportionally less.

CONCLUSIONS.

In choosing the power to be used for tunnel plants, a waterfall or rapid, if either is available, should have first consideration. The chief arguments in favor of water power are as follows: No fuel is required; the cost for attendance and repairs is a minimum; the power is comparatively reliable, obviating losses from interruptions of service. The factor that may prohibit its selection is a high cost of installation and resulting large charge for interest and depreciation per unit of power. This consideration, dependent entirely upon local conditions, usually determines the adoption or rejection of a possible water-power plant. Again, where water power is not obtainable directly at the tunnel plant, if it can be procured from a waterfall in the neighborhood, the essential factors remain the same, except that a means of transmitting the power, such as air or electricity, must be selected and the cost of the transmission system must be included in the cost of installation. Another possible means of obtaining the advantages of water power is current purchased from an established hydroelectric company. In such case, to the price of the power should be added the cost of attendance at the tunnel plant and the interest and depreciation charges on the necessary equipment. Allowance must be made for interruptions to service, which are neither unusual nor avoidable, in long-distance electrical transmission.

The choice of machinery for utilizing water power is also largely governed by local conditions. As high heads, for which impulse wheels are especially adapted, are generally to be found where water power is available for tunnel work, this type of machine is properly chosen in most instances. Turbine wheels may be used where the water is clear or can be settled in a reservoir. The hydraulic compressor, although practically automatic and entailing only a small operating expense, is so costly to build that it is scarcely to be considered, except for plants much larger than those designed for tunnel work.

If water power were not available and electricity were not purchasable, a steam plant would, according to usual practice, be installed. This choice is difficult to understand unless it be attributed

to the supposed unreliability of the gas producer. The usual steam plant for tunnel purposes is, as has been shown, very inefficient in its utilization of the energy of coal and has a fuel consumption rarely less than 4 or 5 pounds per horsepower-hour. As regards cost of installation, the balance is slightly in favor of steam, but not sufficiently so to overcome the disadvantage of higher operating cost.

The producer-gas plant on the other hand, is several times more efficient in its utilization of heat energy, producing a brake horsepower per hour at some plants with as little as 1 pound of coal. With a producer it is also possible to utilize cheaper grades of fuel. The manufacturers of air compressors have recently adapted their machines for use with internal-combustion engines. It would seem, therefore, if a plant using fuel were necessary that the installation of a producer-gas plant under most conditions would be more desirable than a steam plant.

As a means of transmitting power for any great distance the balance is preponderantly in favor of electric transmission at high tension. In tunnel work and over comparatively short distances compressed air is able to compete with it, because the air drills require this form of power for their operation. When it is necessary to obtain power from coal there seems to be a field for producer-gas transmission in the medium distances, where the cost of the line and the power losses in transmission prohibit the use of air, but where the cost of the extra electrical machinery is still not warranted by the saving in cost of line.

AIR COMPRESSORS.

Many factors should be considered in selecting an air compressor. Motive power and capacity have to be considered first; then, perhaps, the type of the compressor. The methods of regulation under varying load likewise deserve attention, as do cooling and devices for removing moisture from the air.

The most familiar types of air compressor consist essentially of a cylinder in which air is compressed by a piston. Automatic devices are provided for admitting and delivering air, and a flywheel is required to keep the piston moving smoothly. When steam or internal-combustion engines are the prime movers they are usually, although not necessarily, incorporated with the compressor, the power and air pistons being connected by a common piston rod or engine shaft. Where water or electricity is employed the compressor usually is belted to the prime motor, the flywheel of the compressor serving as a pulley. But there is a growing demand for the direct-connected electrically driven machine, in which the electric motor forms an integral part of the compressor, the armature serving as a flywheel. Such machines are now supplied by all the leading manufacturers.

Direct-connected air compressors driven by water power are also obtainable; the water wheel serves as a flywheel for the compressor. Air compressors of an entirely different type, operating on the principle of the reverse turbine, have recently been placed on the market. They are especially adapted to take advantage of the high rotative speed of electric motors and steam turbines. Although the Taylor hydraulic system is, strictly speaking, an air compressor, it has been described (pp. 47-49) as a means of utilizing water power and will not be discussed here.

POWER.

Although the kind of motive power selected for a given plant is generally predetermined by local conditions, the amount of power required is worthy of brief discussion. Rix^a states that in compressing air from atmospheric pressure to 90 or 95 pounds,^b 20 brake horsepower must be delivered at the flywheel shaft of a reciprocating compressor for every 100 cubic feet per minute of piston displacement, this figure being deduced as the average result of a number of tests of air-compressor plants, the capacities of almost every kind of compressor being compared with the motive power required. Rix also states that the figures of power required as given in trade catalogues are usually somewhat lower than this value, but such figures are theoretical and do not take into consideration the mechanical or volumetric efficiency of the compressor. The following tables are computed from the catalogue of two leading manufacturers of a popular type of compressor and show the rated brake horsepower per 100 cubic feet of cylinder displacement when the final gage pressure is 100 pounds.

Relation between required brake horsepower and capacity of 2 air compressors, each compressing to 100 pounds per square inch.

Compressor A.		Compressor B.	
Capacity (cubic feet per minute).	Brake horsepower required for each 100 cubic feet of displacement.	Capacity (cubic feet per minute).	Brake horsepower required for each 100 cubic feet of displacement.
144.....	18.7	248.....	19.3
247.....	18.6	338.....	19.2
372.....	18.4	537.....	18.1
534.....	18.3	680.....	18.1
704.....	18.1	873.....	18.0
1,051.....	18.0	1,056.....	18.0
1,312.....	17.8	1,188.....	18.0
1,662.....	17.7	1,414.....	17.9
2,381.....	17.7	1,846.....	17.9

^a Rix, E. A., Compressed-air calculations: Address before the mining association of the University of California, Feb. 19, 1908; reprinted in *Compressed Air Mag.*, June, 1908, p. 4894.

^b Throughout this report when air pressure is mentioned the figures given are those above atmosphere; that is, gage pressure.

It will be observed that the table supports the statement made by Rix, and that even in spite of the increased final pressure, the values are somewhat less than the one he proposes. They also show that in machines of large capacity, proportionally less power is required.

The following table, based on published figures, shows the amount of power required or provided per 100 cubic feet of free air actually compressed at several turbocompressor plants:

Power consumption of turbocompressors.

Pressure, pounds per square inch.	Capacity, cubic feet of free air per minute.	Rated horsepower of motor or engine per 100 cubic feet of free air when compressing to stated pressure.	Actual horsepower required to compress 100 cubic feet of free air per minute to stated pressure.
80.....	4,600	21.8	
118.....	21,250	18.8	 17.0
135.....	20,000		 18.5
170.....	22,000	18.2	

CAPACITY.

The capacity of compressors is rated in free air,* and in reciprocating machines is based equally on speed and on the piston displacement—the number of cubic feet of cylinder space swept by the piston each minute at the given speed. However, there are unavoidable losses in volume because of clearance, piston speed, leakage, and expansion, the sum of which may amount to as much as 30 per cent of the rated capacity in a single-stage compressor at 100-pound pressure. The capacity of turbocompressors is based on the volume of free air drawn into the intake per minute. Although some of the more carefully designed reciprocating compressors may give a volumetric efficiency as high as 90 per cent, for compressors such as are customarily employed in power plants for tunnels 80 per cent is more likely to be the figure, and although the tables in manufacturers' catalogues of air drills are in the main fairly accurate for new drills, the air consumption may be greatly augmented as parts become worn. Moreover, provision must be made for pipe-line leakage and for air required by drill sharpeners, blacksmith forges, and an extra small drill, if one is used, for blocking and trimming. It is therefore most desirable to have the air compressors considerably oversized as based on catalogue rating, and in tunnel practice the oversize usually ranges from 100 to 150 per cent. The following table shows a comparison between the rated compressor capacity and the design-

* Free air is air at atmospheric (14.7 pounds) pressure and at a temperature of 60° F.

nated air consumption for the drills employed at several tunnels and adits:

Relation between compressor capacity and air consumption of drills.

Tunnel.	Catalogue figures for compressor.		Drills.		Altitude.	Oversize of compressor. ^a
	Speed.	Capacity.	Number in heading.	Stated air consumption.		
	<i>R. p. m.</i>	<i>Cubic feet per minute.</i>		<i>Cubic feet per minute.</i>	<i>Feet.</i>	<i>Per cent.¹</i>
Carter.....	150	868	2	230	9,000	280
Laramie-Poudre.....	165	902	3	250	8,000	140
Elizabeth Lake.....	190	736	3	185	3,000	300
Lucania.....	150	544	3	250	8,000	120
Marshall-Russell.....	175	487	2	200	8,000	140
Mission.....	190	247	1	100	1,200	150
Rawley.....	175	427	2	190	10,000	120
Snake Creek.....	165	680	2	300	6,000	125
Strawberry.....	175	427	2	300	7,000	40

^a Not including drill sharpeners, forges, or leakage in pipe lines.

The loss of effective compressor capacity through leaky pipe lines is not fully realized at many plants and no steps are taken either to determine the amount of this waste or to prevent it. If the pipe lines are constructed with great care and covered so as to protect them from accident or from extremes of temperature, the loss by leakage may be almost negligible, but if they are not well built or if the surface lines are exposed to injury from numerous causes—not the least being diurnal and seasonal variation in temperature—and those underground are liable to be struck by falling rock or derailed cars, the leakage is likely to be considerable, and the lines should be tested at short intervals in order that the loss may be ascertained and stopped. If reciprocating compressors, driven either by steam or water power, are used, the leakage can be easily ascertained by closing all of the outlet pipes from the line and noting the number of strokes per minute necessary to maintain the desired pressure; but if turbocompressors are used, unless the output at different speeds and pressures has been carefully determined and tabulated, the leakage can best be ascertained by stopping the compressor when the receiver and lines are filled, allowing the pressure to drop to, say, 50 per cent, then starting the machine and noting the time required for the pressure to reach the original point. Although this course does not give exact results, still it will furnish a useful index to the rate of leakage, and is so simple that one might think it ought to be in general use. Unfortunately, the habit of workmen, especially the “chain gang,” seems to be to assume that all air lines are much freer from leaks than they really are. A few years ago at a large western mine, where a great number of drills, drill sharpeners, and pumps

were driven by compressed air, when the required pressure could not be maintained, bids for a new and expensive compressor were asked for, but the management thought of testing the pipe lines by the method indicated above, and it was discovered that 1,100 horsepower was being used to supply the loss by leakage. In this case the "chain gang," instead of the machine shop, "got busy" and in a week the waste of air was so reduced that instead of another compressor being bought one of the largest of those in use was shut down.

TYPES.

Reciprocating air compressors may be divided into two general types: "Straight line," sometimes called "tandem," and duplex. Either type may be single stage if the air reaches its final pressure in one cylinder, or multistage if the compression is completed in a second, third, or even a fourth cylinder.

STRAIGHT-LINE COMPRESSORS.

In a tandem compressor, if it be driven by steam or an internal-combustion engine, the power and air cylinders have a common piston rod and the power is applied in a straight line. The flywheels, of which there are usually two, may be at either end or between the cylinders and are connected to the piston rod by a crosshead and ordinary connecting rods. If the tandem compressor be driven by electricity or water, practically the only change is the omission of the power cylinder.

DUPLEX COMPRESSORS.

A duplex compressor consists of two tandem compressors placed side by side and having between them a flywheel on a common shaft. The two sides are connected to the flywheel shaft by cranks set at 90°, so that when one side is encountering maximum resistance the other is working under the lightest load. Many different combinations are possible. The steam cylinders * may or may not be compounded, and the air cylinders may be single stage or multistage. Again the steam cylinders may be omitted and the power transmitted to the machine by a belt or by a direct-connected motor.

TURBOCOMPRESSORS.

The working principle of a turbocompressor (fig. 5) is that of a reversed turbine, air, instead of water or steam, being the fluid acted upon. Such a compressor consists essentially of a revolving impeller, not unlike that of some centrifugal fans, surrounded by stationary discharge vanes supported in a suitable casing. The

* Internal-combustion engines have not as yet been applied to this type of compressor.

discharge vanes recover the major part of the energy that exists as velocity in the air leaving the impeller, which is roughly almost one-half of the total energy supplied from the driving machine, and convert this velocity into available pressure. In a centrifugal fan, which has no such vanes, this energy is lost as heat produced by eddies and friction; hence it is not difficult to see why a turbocompressor is more efficient. Single-stage turbocompressors are employed chiefly in metallurgical plants, at blast furnaces and cupolas, and could be used for mine ventilation; but, if a high pressure is required, such as that needed by rock drills and other pneumatic machinery, several impeller units are mounted in series on a common shaft within a common casing, the air from the first set of discharge vanes going to the intake of the second impeller, and so on. Compressors producing a pressure of 170 pounds and having as many as 29 stages have been constructed, but if so many stages are employed the impellers are usually mounted in groups of four to ten.

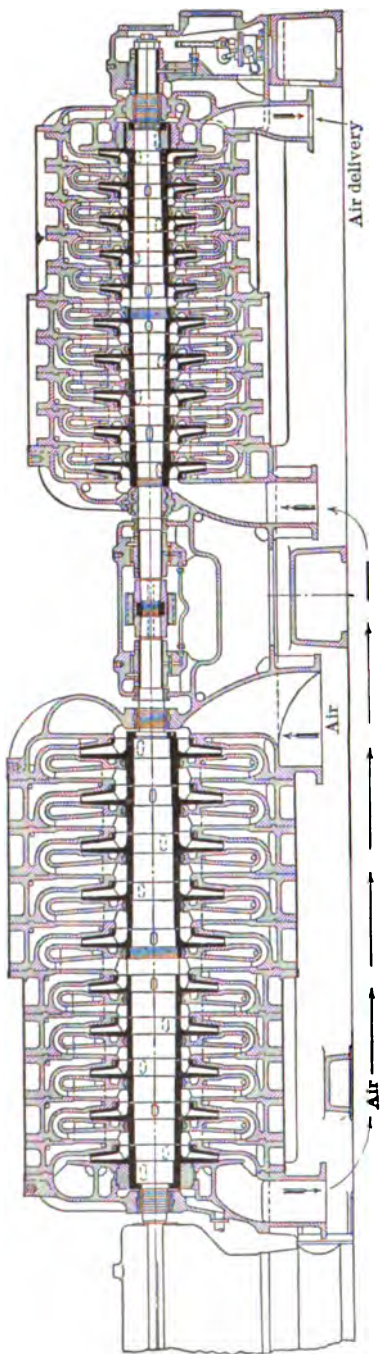


FIGURE 5.—Section of a turbocompressor.

Turbocompressors are just beginning to be made in this country, but they have been in use for several years in Germany, where their design and manufacture have already reached a high degree of perfection. The first large turbocompressor was built in 1909 for the Reden mines near Swarbrucken. It is driven by a 1,000-horsepower, mixed-pressure steam turbine at 4,200 revolutions per minute, and compresses 4,600 cubic feet of free air per minute to a gage pressure of 90 pounds to the square inch. Rather recently six motor-driven

compresses 4,600 cubic feet of free air per minute to a gage pressure of 90 pounds to the square inch. Rather recently six motor-driven

compressors of this kind were built in Germany to supply power for the Rand mines in South Africa, each of them being driven by two 2,000-horsepower synchronous motors running at 3,000 revolutions per minute. The compressors have a rated capacity of compressing 21,250 cubic feet of free air per minute at 68° F. to 118 pounds pressure per square inch. In a test, when compressing 23,750 cubic feet of free air per minute to 100 pounds pressure, the energy of consumption per 100 cubic feet of free air was 17 horsepower, and the highest isothermal efficiency obtained was 67.04 per cent. The first large turbocompressor built in this country went into service in May, 1911, and has been in continuous operation ever since; it is driven by a steam turbine at 4,700 revolutions per minute and has a capacity of 3,500 cubic feet of free air per minute delivered at a pressure of 105 pounds per square inch.

For material relative to other of the numerous makes of air compressors the reader is referred to the trade catalogues of the manufacturers, who will be glad to supply information and to render assistance in the selection of a compressor.

COMPARISONS.

The chief advantages of the straight-line compressor are strength, simplicity, compactness, and ease of installation. It is usually self-contained, being mounted on a single bedplate, and requires relatively inexpensive foundation. The frictional losses in a good machine of this type are not large, and it may have a fairly good power economy at or near full load with moderate pressures. These features make it advantageous for less accessible plants or those that are somewhat temporary.

A great advantage of the duplex type is the facility with which either steam or air cylinders may be "compounded" without increasing materially the number of parts. Hence the duplex type can take advantage of the great saving in power resulting from compound steam cylinders as well as of the economy resulting from two-stage air compression. Practical experience with the two types of machines fully confirms the theoretical investigations of their comparative efficiency, and carefully conducted tests extending over long periods of time have established the economical superiority of the duplex type. This type, also, if properly designed, shows little, if any, greater mechanical losses through friction, etc., than the straight line, and it is much more easily regulated under varying loads. Most duplex compressors are now made with a substantial subbase that gives them a strength and rigidity comparable with those of the straight-line type, reduces the outlay for foundation, and thus meets some of the conditions that have until recently been so much in favor of that type. The result is that, with perhaps

a half dozen exceptions, the air compressors at the tunnel plants examined were of the duplex type.

The entire absence of valves, reciprocating parts, and sliding friction in the turbocompressor, together with its freedom from vibration, its high capacity in proportion to weight and to floor space occupied, and its ability to take advantage of the high rotative speeds of electric motors and steam turbines, is certain to bring it into general use. Using live steam, condensing or non-condensing turbine-engine turbocompressor units can compete successfully with the highest grades of reciprocating-engine compressor plants; moreover they can successfully utilize exhaust steam from engines, pumps, or other apparatus, which forms one of the cheapest possible sources of power, because, by utilizing steam that would otherwise go to waste, the cost of fuel is practically nothing.

Another advantage, and one that may easily be overlooked, is the fact that the turbocompressor practically eliminates the danger of explosions in air receivers and pipe lines. The cylinders of piston compressors must be lubricated, and the lubricating oil, which becomes finely divided, has ample opportunity to mix with the air being compressed and to be carried with it into the receiver, but in the turbocompressor only the bearings require lubrication, and to these the air being compressed has no access.

When an electric motor or water wheel is the source of power, the turbocompressor may be easily connected to either without loss from speed reduction and friction, a feature that renders the combination highly desirable. The turbocompressor is readily adapted to automatic control and may be regulated for the delivery of a constant volume or constant pressure, as required. Its efficiency is maintained over a wide range of load within a few per cent of the maximum, and the efficiency does not decrease with continued service.

REGULATION.

STEAM-DRIVEN COMPRESSORS.

Although changing the load changes the speed of any steam-driven machine, such change, especially with high air and steam pressure, is to the advantage of the straight-line compressor, because this type will not run satisfactorily at low speeds on account of the insufficient momentum of the flywheel. To avoid stoppage, either the steam cut-off must be lengthened, involving a loss of steam as the machine speeds up to supply more air, or the steam must not be decreased below a fixed limit, and when the demand for compressed air falls below that supplied regularly by the machine the excess must be permitted to escape through a safety valve. Either of these operations entails loss of power. For this reason the straight-line compressor can not operate economically much below 40 per cent of full load.

In the matter of regulation of steam-driven compressors the duplex has an unquestioned advantage over the straight-line type. The quartered cranks, in addition to minimizing strains and reducing extremes, enable one cylinder to come to the help of the other just at the time when that help is most beneficial. There can be no dead center and the machine will run so slowly as hardly to turn over if the compressed air in the receiver is not being drawn upon, and will speed up rapidly as the demand for air increases, no change in the cut-off being necessary. The duplex machine, therefore, has about the same steam economy over the full range of load, without any loss of compressed air at the safety valve.

The regulation of the turbocompressor when steam-turbine driven is merely a matter of controlling the steam admitted to the turbine.

WATER-DRIVEN COMPRESSORS.

Compressors driven by impulse wheels may be regulated by several devices, including the deflecting nozzle, the needle nozzle, and the cut-off. The deflecting nozzle is provided with a ball-and-socket joint and is controlled by air-receiver pressure in such a manner that a part of the stream of water may be shifted on or off the buckets of the wheel, thus increasing or decreasing the power developed. The nozzle may be stationary and a steel plate may be made to deflect the stream of water. The needle nozzle is merely a discharge valve in which the movement of a conical needle diminishes or increases the amount of water passing through. The cut-off, also, regulates the quantity of water discharged, a tight-fitting plate being shifted across the nozzle tip. The deflecting devices can vary the power rapidly, but, of course, waste water; the reverse is true of the other devices. For tunnel plants, however, as water economy is rarely an essential consideration and variations in load are frequent and sudden, the deflecting devices are most suitable.

ELECTRICALLY DRIVEN COMPRESSORS.

In any reciprocating machine the volume of air compressed varies with the number of piston strokes per minute. In the turbine compressor this volume depends on the rotative speed. In electrically driven reciprocating compressors, whether direct-connected or belted to a motor, the speed has to be reasonably constant and can not be varied to meet fluctuations in the demand for air, and as economy obviously forbids the discharge of excess compressed air through a safety valve, "unloaders" must be provided to overcome the difficulty.

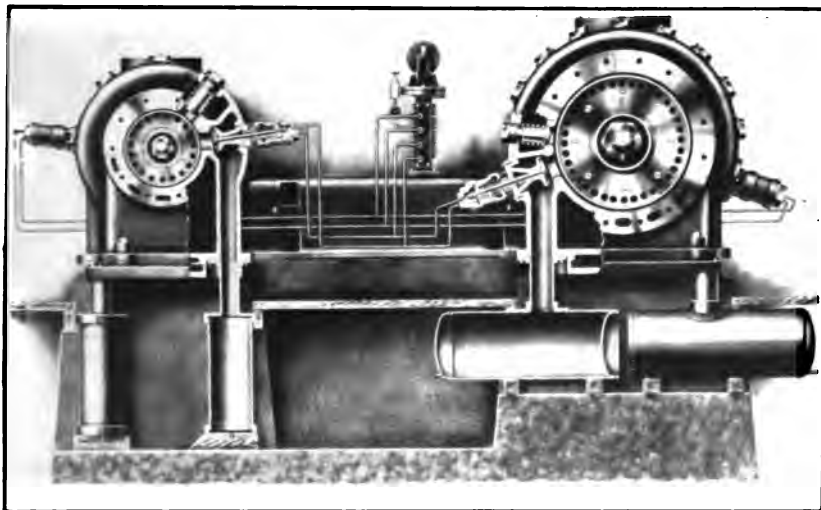
Unloaders.—The more common type of unloader limits the amount of air admitted to the compressor. A valve in the free-air intake pipe is controlled by the pressure in the air receiver, so as to throttle

the admission of air when the load is light, and allows more to enter when the demand for air increases. The device may be employed successfully with turbocompressors, but with reciprocating machines it has objections, because when the machines are running with a partly throttled inlet, the air drawn into the cylinder, being of smaller volume, is rarefied and on the return stroke of the piston is compressed through a greater range of pressure, so that the temperatures are higher than ordinary and may reach unsafe limits, especially if the terminal pressure is great. This feature is not so important with turbocompressors, because the temperatures, on account of the repeated cooling between stages, never become so high as in reciprocating machines.

On some piston compressors the unloader employed is of almost exactly opposite type, consisting of a device for holding the intake valves open whenever the air pressure reaches a predetermined point; it avoids the high temperatures developed with the type previously mentioned.

In one type of unloader for reciprocating machines the excess air is forced automatically into clearance tanks, the process being controlled by a predetermined receiver air pressure. A view of this device, partly in section, is shown in Plate II, A. Under normal full load the controller is inoperative, but at less than full capacity some of the compressed air is forced into the tanks instead of through the discharge valve, thus reducing the output. On the return stroke this air expands and returns its stored energy to the piston. There are 8 tanks in all, and five equal and successive unloading stages are possible by throwing in respectively 1, 2, 4, 6, or all of the tanks. The regulation is said to be unaccompanied by shock from sudden variations in load, and heating from excessive compression ratios is avoided. In fact, as there is a little radiation from the clearance tanks, the air is probably returned to the cylinder slightly cooler than when it left.

Another method of unloading is by holding open the discharge valves of the compressor, and permitting compressed air instead of free air to fill the cylinder as the piston retreats so that the pressure on both sides of the piston is balanced. Although this procedure unloads the compressor completely it has a serious drawback. As the load is resumed, the balance of pressure is disturbed, one side of the piston being subjected to something less than atmospheric pressure, the opposite side being exposed to the full pressure of air in the receiver, the difference in pressure being thrown on the piston instantly and maintained throughout the entire stroke. The resulting heavy strain on the compressor prohibits the use of this unloader except in the smaller sizes. Still another type releases the partly



A. ELECTRICALLY DRIVEN AIR COMPRESSOR WITH FIVE-STEP CLEARANCE REGULATOR.



B. VENTILATING BLOWER USED AT LOS ANGELES AQUEDUCT.

compressed air during its passage from the low to the high pressure cylinders, but little can be said for this method except that it is not so wasteful as releasing high-pressure air.

HEAT FACTORS.

HEAT PRODUCED.

Heat is produced during the compression of air, the rise in temperature being largely dependent upon the ratio of the initial to the final pressure. For instance, if air at 60° F. be compressed in a single stroke from atmospheric pressure to 100 pounds gage, the temperature attained would be 485° F., assuming no loss by radiation during the process. On the other hand, under the same conditions, if the final pressure were only 25-pounds gage, the air would be heated to only 233° F., and if it were then cooled again to 60° F. and further compressed from 25 pounds to 100 pounds gage the final temperature would be approximately 250° F. The effect of the increase in temperature is to cause the air to expand to a larger volume; hence more work is required to compress it. If the air could be used at once to operate a motor, before any of the heat escapes through radiation, etc., this work could be obtained again from the air; but as in mining work the heat is almost without exception entirely dissipated in the pipe line before the air reaches the drills, the production of heat during compression entails a serious loss of power.

DANGERS OF HIGH TEMPERATURES.

Aside from the item of power waste, the temperature reached during compression has an important bearing on the subject of explosions in air lines. It can readily be imagined that if the discharge valves are not working properly and some of the highly heated compressed air is allowed to reenter the cylinder with the fresh intake air, compression may begin at a temperature much higher than normal. In this case, even with two-stage machines, the final temperature of the compressed air may be gradually built up from 250° to 500°, 600°, or even higher. The temperature is often high enough to volatilize lubricating oil, the vapors of which, mingling with the air, may form an explosive mixture. If the temperature then becomes high enough to ignite the mixture an explosion must result.

REMOVAL OF HEAT.

The ideal way to prevent the evil effects of heat would be to devise some means of removing it from the air as fast as produced during compression. This is fairly approximated in turbocompressors. With piston compressors such a course is unfortunately impossible

in practice, but various devices partly accomplish the result. A familiar one is to surround the cylinder with a jacket of cooling water, the piston also being sometimes cooled in this way. But when one considers that air is a very poor conductor of heat, and that at the time when it is hottest it occupies only the minimum volume in one end of the cylinder, and even then for only a short time, it will readily be seen that this method can not be very effective. In some modern compressors the inlet valves are placed in the piston and the discharge valves in the ends of the cylinders instead of in the heads, thus permitting the heads to be fully water jacketed, a practice that is to be most highly commended. As water jacketing is the only means used to cool the air during single-stage compression, such machines are not economical of power.

In two-stage compressors, however, a part of the heat is actually removed during the process of compression by means of intercoolers provided for this purpose. The air is only partly compressed in the first cylinder, the heat produced being practically all removed during the passage of the air through an intercooler on its way to the second cylinder. By removing the heat in the intercooler, the final temperature of the air is kept much lower than with single-stage compression; hence there is less expansion of the air to be overcome, resulting in a consequent saving of power. In a properly designed two-stage machine compressing to 100 pounds gage this saving of power is approximately 13 per cent, and it increases with the higher terminal pressures. If the pressure is less than 80 pounds the saving is hardly great enough to be a serious consideration and single-stage machines are customarily employed in such cases, but for pressure higher than 100 pounds, two-stage compression is imperative because of the high temperatures that are otherwise produced. As shown by the following table, the pressure ordinarily employed in tunnel plants ranges from 80 to 120 pounds, averaging about 100 pounds:

Compressed-air pressures at different tunnel plants.

Plant.	Pounds.	Plant.	Pounds.	Plant.	Pounds.
Carter.....	112	Marshall-Russell.....	110	Roosevelt.....	110
Central.....	120	Mission.....	100	Siwatch.....	80
Gold Links.....	100	Moodna.....	95 to 100	Snake Creek.....	110
Gunnison.....	90	Newhouse.....	110	Stilwell.....	100
Laramie-Poudre.....	120	Nisqually.....	90 to 95	Strawberry.....	85
Mauch Chunk.....	100	Rawley.....	100	Utah Metals.....	110
Los Angeles Aqueduct..	100	Raymond.....	90	Walkill.....	110
Lucania.....	115	Rondout.....	100	Yak.....	90
				Average.....	102

Because of the many stages required with turbocompressors when delivering air for use in drilling, the difference between the pressures of the air on entering and leaving any one stage is extremely small

compared with the pressure of two-stage reciprocating machines. Hence the resulting increase of temperature in any one step is not great, and the comparatively small amount of heat generated can be effectively removed by the use of a suitably designed water jacket. Some idea of the efficiency obtainable with such a cooling system may be had from considering the fact that the air delivered into the receiver at 105 pounds pressure from the turbocompressor mentioned on page 69, has a temperature of only 120° F.

INTERCOOLING.

The efficiency of two-stage compression is dependent upon the intercooler. It is an essential part of this type of machine and usually consists of a shell, generally cylindrical in shape, containing a number of tubes, similar to those in a tubular boiler, through which cold water is made to circulate (fig. 6). The heated air from the low-

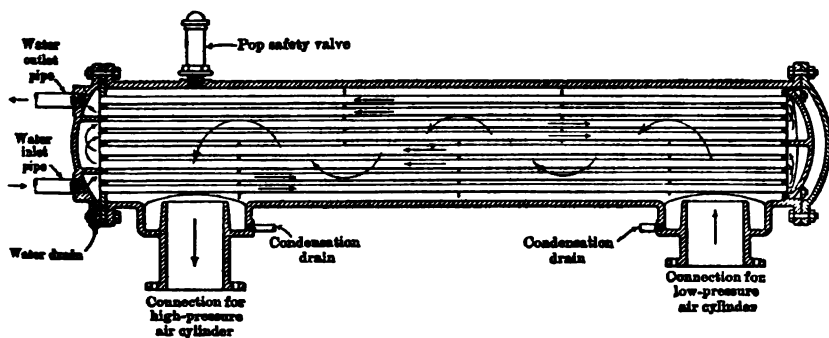


FIGURE 6.—Section of an intercooler.

pressure cylinder enters near one end, passes through the nest of tubes, its passage being obstructed by baffle plates to insure the maximum contact between the air and the cooling surface, and is delivered at much lower temperature to the high-pressure cylinder at the other end. The success of the intercooler depends upon several considerations. In order that the least dependence need be placed upon the heat conductivity of the air itself, which is notably poor, the intercooler must subdivide the air completely and insure that the maximum amount is thrown in contact with the cooling surfaces. This is accomplished by properly spaced water tubes and baffle plates. At the same time the cross section of the cooler must not be too small, lest the velocity of the air past the cooling surface be so great that sufficient time be not allowed for the water to absorb all the heat. It is desirable also to have the water and air flow in opposite directions in order that the final cooling of the air may be effected by the entering, and consequently the coldest, water. Theoretically the cooling surface should be sufficient to absorb all the heat in the air passed

over it, and thus reduce the temperature to that at which the air entered the low-pressure cylinder, but even in good practice, owing possibly to mechanical difficulties, intercoolers usually fail to do this within 5 to 10 or even 40 degrees.

MOISTURE FACTORS.

The intercooler assists also in removing water from the air. Normal air always contains some water vapor, the amount absorbed being determined by the temperature and pressure. In the air compressor the vapor would be condensed out of the air as the volume is reduced were it not that the increase in temperature more than offsets the effect of the decrease in volume, so that actually the vapor-holding capacity of the confined air is slightly increased. As the air passes through the intercooler the temperature is lowered greatly and the air gives up moisture which is precipitated in a finely divided state and does not settle for some time; hence only a part of it can be collected in the intercooler and drawn off through the drains provided unless the intercooler is so made as to act also as a mechanical separator. Some moisture is swept along with the air to the higher pressure cylinder and revaporized by the temperature therein.

ACCESSORIES.

PRECOOLERS.

Cooling the air before its admission into the air compressor may assist in removing some water from it, and there are a number of precooling devices. One^a improvised device consists simply of a number of odd pipes set between two wooden boxes. The pipes are wrapped with cloth; arrangement is made to have water drip on them constantly, so that the air is cooled by evaporation as it is drawn through the pipes from one box to the other on its way to the compressor intake. At a plant in Johannesburg the air for the compressors is obtained through a subway leading to the center of a building with air-tight roof and floors, and with walls of constantly wetted cocoa matting. At another plant the sides and roof of a similar structure were covered with burlap, both inside and out. A cooler of this type also filters the dust and grit that might seriously injure the cylinder or piston of the compressor, and can not be too strongly recommended in dusty situations. Precooling the air, and thus reducing the volume at a given pressure, increases the capacity of the compressor; hence a greater weight of air is drawn in and compressed at each stroke.

^a Described in *Engineering and Mining Journal*, issue of Nov. 27, 1909, p. 1081.

AFTERCOOLER.

The aftercooler^a is not generally employed in tunnel plants. In cooling the air as this comes from the high-pressure cylinder it precipitates some of the water vapor and also reduces the volume of the air and practically eliminates the danger of explosion in the air line. The water vapor is usually so finely divided that all of it does not at once fall out because of the reduced temperature, part being swept along with the air and deposited both in the air receiver and in the pipe line. There should therefore be provision for draining this water. The reduction in volume is somewhat problematical and is probably not a serious consideration.

AIR RECEIVERS.

The air receiver, a cylindrical shell of steel provided with inlet and outlet pipes and usually a safety valve, is supposed, according to the popular notion, to store, cool, and dry the air and to equalize irregularities in its production and use. In actual practice, however, the receiver probably does little more than partly to equalize irregularities. As the receiver ordinarily used in tunnel plants rarely has a capacity greater than one minute's run of the compressor it can not possibly furnish much storage space, and as the air in the receiver is being renewed each minute, if the compressor is in operation, its velocity through the receiver must be enough to prevent much cooling. The air near the shell loses some heat, of course, but this loss is small compared to the heat in the mass of the air nearer the center of the receiver, so that the air leaves with a temperature only slightly, if at all, less than that at which it entered. Furthermore, as there is practically no cooling of the air there can not be much precipitation of water vapor. In practice, although most air receivers are provided with a drain of some sort, only a ridiculously small quantity of water is ever drawn off. On the other hand, some receivers have actually become combustion chambers by reason of oil and grease collecting on the inside of the shell and being ignited when the temperature of the air became high enough. However, the receiver and the pipe line, which may be considered as supplying auxiliary storage space, assist greatly in equalizing the pulsations of the air delivered from the compressor and of that used by the drills, and in this way it reduces the strains on the compressor. By regulating the flow it does not permit the air to attain a high velocity in the pipes, and hence power is saved, as the friction losses increase greatly with the velocity. To obtain the maximum benefit from this factor a second receiver is often installed

^a In design and principle the aftercooler is practically the same as an intercooler, and when used it is generally placed between the compressor and the air receiver.

as near as possible to the place where the air is to be used. The second receiver assists materially in maintaining a steadier air pressure at the drills and makes it possible to use a smaller pipe line. A tubular boiler, which may often be bought cheaply at second hand, makes an excellent receiver and a very efficient cooler. With a vertical tubular boiler it is necessary only to remove the fire and ash doors to provide for ventilation, whereas a horizontal tubular boiler should be placed on an incline sufficiently steep to insure a rapid draft of outside air through the flues.

DRAINS.

As practically the entire cooling of the air after it has left the compressor takes place in the pipe line, most of the water is precipitated there and causes serious inconvenience in several ways. During cool weather, through continued deposition and freezing, the pipe line may become closed altogether or so restricted as to cause serious drop in pressure or loss of power; or the water getting into the exhaust from the drills may prevent their operation through freezing at the low temperature of the expanded air. The obvious remedy is to remove the water, which is done by draining the low places in the line where the water collects. This can be accomplished automatically by the use of any good float-design steam trap, but if the pipe is exposed to low temperatures the trap should be placed in a small pit or otherwise protected to prevent freezing. If necessary further provision for the elimination of moisture from the compressed air and water from the pipes can be had by placing in the line any high-class standard steam separator, fitted with an automatic trap, as described above. For equally satisfactory results a larger size of separator is, however, necessary than for steam, as the carrying power of the more dense compressed air is greater than that of steam.

SUMMARY.

A brief summary of the factors that enter into the problem of selecting an air compressor follows. The power required for both reciprocating and turbine machines is approximately 18 to 20 brake horsepower for every 100 cubic feet of free air compressed to 100 pounds gage. The values given in trade catalogues for reciprocating compressors are generally a little below this figure, but it is a safe one to use in estimates. Such compressors ordinarily have a volumetric efficiency of approximately 80 per cent, and as they are rated on the basis of free air and as it is necessary to make allowance for loss due to clearance, etc., provision for increased air consumption above the catalogue rating for drills as they become worn, and for air used in sharpening machines and forges, must also be made with either reciprocating or turbine machines; hence it is advisable to select an air com-

pressor considerably oversize. In practice the oversize, based upon drills only, ordinarily ranges from 100 to 150 per cent.

Of the two types of reciprocating compressors the duplex is preferable to the usual straight line, in spite of the latter's simplicity and easier installation, because of the former's more economical and efficient use of power and the facility of its regulation, especially when driven by steam under high pressure.

As the air pressure at tunnel plants is rarely below 80 pounds, in three out of every four being 100 pounds or greater, two-stage compression is desirable because of its economy of power. Indeed, such compression may be imperative because of the high temperatures that might otherwise be attained.

Although the manufacture of turbocompressors is just beginning in this country, they possess a number of advantages, especially for use with steam turbines and other rotary engines operating at high speeds, and will doubtless be more generally used in the future. Their development should therefore be closely watched. Steam-driven compressors are regulated by varying their speed; but, as in some power-driven machines, the speed is necessarily constant, other means, of which the throttle inlet and the clearance controllers are the two most used, must be provided for that purpose. Heat is produced during compression and, by expanding the air, causes loss of power. Some of this loss is obviated in stage compression by removing the heat during its passage through an intercooler between the cylinders. The numerous stages in the turbine machine enable this heat to be removed effectively by water-jacketing. Another evil attributable to this heat is the danger from the explosion of volatilized lubricating oils; but in the turbine machine this danger is eliminated because there are no sliding surfaces to require lubrication. Among the accessories that are designed to prevent or neutralize the effects of heat in piston machines are the precooler, the intercooler, the after-cooler, and the air receiver. The air receiver also equalizes the pulsations of the air and reduces friction losses. The devices mentioned assist also in freeing the air from water which often causes serious inconvenience. The major part of the water is apt to be deposited in the pipe line, however, if this is the coldest part of the system, and provision must be made for its removal.

VENTILATION EQUIPMENT.

MACHINERY.

Either "blowers" or fans are employed ordinarily for ventilating tunnels and adits. In every revolution of a positive blower a certain quantity of air is trapped between the impellers and the inclosing casing and has no means of escape except through the discharge pipe,

as is indicated in figure 7. For this reason blowers are often styled "pressure" blowers and "positive-blast" machines. Plate II, *B*, shows one of these blowers in operation on the Los Angeles Aqueduct.

If fans are employed in tunnel ventilation they are almost without exception of the centrifugal type, the disk form, similar to the ordinary desk fan, being rarely used. In the centrifugal fan the air enters near the center in a direction approximately parallel to the axis of the shaft and is forced by the centrifugal action of the rapidly revolving blades toward their periphery, where it is discharged. This type has many modifications designed to prevent loss of efficiency through friction as the air strikes the back plate and changes direction or to prevent eddies due to the greater density of the air at that point caused by its momentum upon entering the fan.

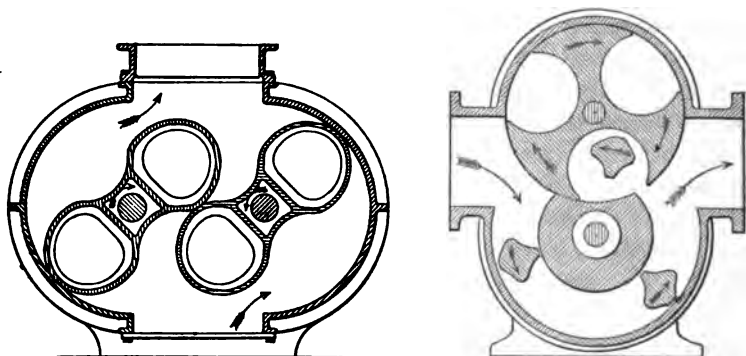


FIGURE 7.—Cross sections of pressure blowers, indicating the course of the air.

Turbocompressors in which by one, two, or even several stages air can be delivered at any required pressure have been employed as blowers for blast-furnace and foundry work at a number of places. The capacities of those manufactured for this purpose thus far are too great for the requirements of tunnel work, but their greater efficiency as compared with centrifugal fans, and the possibility of designing them to obtain any required pressure, will doubtless soon lead to their being made in sizes suitable for tunnel work, in which they should have a large field.

At one tunnel vitiated air was removed from the heading by the use of a jet of highly compressed air which was directed into the ventilating pipe, but this method in addition to being expensive is inadequate, and is, therefore, not to be advised, except as a temporary expedient and for short distances. On short levels and crosscuts, however, or on larger work pending the installation of more expensive and efficient machinery, jet blowers can often be used to good ad-

vantage. They can be operated by either compressed air or water under pressure and, though far from being as efficient as the mechanical types of ventilating machinery, will in many cases perform an extremely useful function. Jet blowers can frequently be used to move large volumes of air for short distances against low frictional resistances with good results, and their extreme economy in first cost makes them an excellent accessory in preliminary work.

DIRECTION OF CURRENT.

By a proper adjustment of the ventilating pipe the fan or blower ordinarily installed for tunnel work may be made to exhaust the air from or deliver it to the heading. One of the chief advantages of exhausting the air is that the dangerous gases and smoke produced in blasting are removed from the tunnel or adit promptly, and it is therefore unnecessary for the workmen to pass through the thick bank of smoke that would otherwise travel very slowly to the portal. On the other hand, when fresh air is blown in, it passes very much faster through the pipe and is cooler and fresher than if it has worked its way slowly in through the tunnel or adit and become heated from

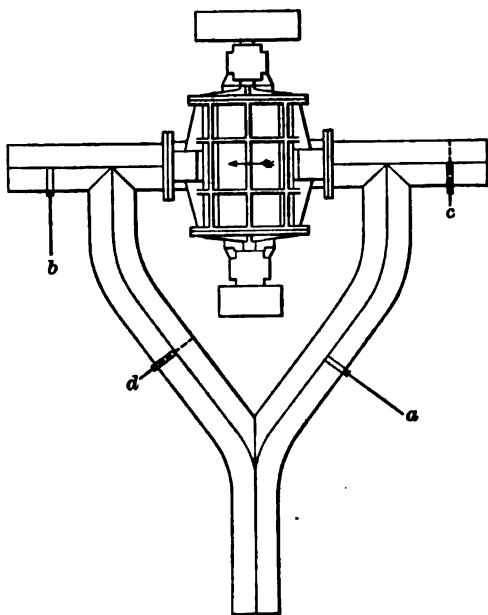


FIGURE 8.—Arrangement of gates and pipes for changing direction of ventilating current.

contact with the walls and contaminated by odors from the floor. The men, therefore, feel more comfortable and are able to do better work when this method is employed. The advantages of both methods, however, may be readily obtained by an arrangement of pipes similar in principle to the one shown in figure 8, which permits the air to be exhausted for a few minutes after blasting by opening gates *a* and *b* and closing *c* and *d*, assuming the current through the fan or blower to be in the direction of the arrow, while at other times by reversing this arrangement air may be forced into the heading. Table 5 shows the direction of the air current at various tunnels visited, from which it may be seen that, almost without exception, it is customary to exhaust the smoke, after blasting at least,

although at many places the ventilating current is reversed at other times. This arrangement is reported as giving excellent results, and its use is strongly recommended by the authors.

TABLE 5.—*Direction of air current at various tunnels.*

Tunnel.	Ordinary current.	Current after shooting.	Tunnel.	Ordinary current.	Current after shooting.
Carter.....	Exhaust.....	Exhaust.	Marshall-Russell.	Exhaust.....	Exhaust.
Central.....	do.....	Do.	Mission.....	Pressure.....	Exhaust $\frac{1}{2}$ to 1 hour.
Gold Links.....	do.....	Do.	Newhouse.....	Exhaust.....	Exhaust.
Gunnison, east portal.	do.....	Do.	Nisqually.....	do.....	Do.
Gunnison, West Portal.	Pressure.....	Exhaust for 2 hours.	Rawley.....	do ^a	Do.
Laramie.....	Exhaust.....	Exhaust.	Raymond.....	Pressure.....	Exhaust for 2 hours.
Lausanne Pandre.	Pressure.....	Pressure.	Rondout.....	do.....	Exhaust "for a while."
Los Angeles Aqueduct:			Roosevelt.....	Exhaust.....	Exhaust.
Elizabeth Lake.....	do.....	Exhaust 20 to 25 minutes.	Siwatch.....	do.....	Do.
Little Lake.....	do.....	Exhaust for 1 hour.	Snake Creek.....	do.....	Do.
Grapevine.....	do.....	Exhaust $\frac{1}{2}$ to 1 hour.	Stillwell.....	do.....	Do.
Lucania.....	Exhaust.....	Exhaust.	Strawberry.....	do.....	Do.
			Utah Metals.....	do.....	Do.
			Walkill.....	Pressure.....	Do.
			Yak.....	Exhaust.....	Do.

^a Intermittent.

CAPACITY.

There is no absolute method for determining the quantity of air needed to renew that vitiated by the respiration of men and animals working in tunnels. For coal mines many States have provided a legal minimum that ranges from 85 to 300 cubic feet per minute for each man and from 300 to 600 cubic feet for each animal. These figures, however, have practically no bearing on tunnel work, because in coal mines a much larger volume of air than that actually needed by the men must be supplied in order to dilute and render harmless the inflammable and dangerous gases given off from the coal. In many States the laws provide that even the requirements mentioned may be increased at the discretion of the mine inspector. Conditions in metal mines, on the other hand, are more closely akin to those in tunnels, but, unfortunately, most existing legislation merely stipulates that the ventilation must be "adequate." Arizona specifies the amount considered harmful by chemical analysis.

Richards^a considers that the following air quantities are sufficient for proper ventilation in metal mining work:

Per light, 1 cubic foot per minute.

Per man, 25 cubic feet per minute.

Per animal, 75 cubic feet per minute.

The mining-regulations committee of the Transvaal, on the other hand, provide (for metal mines) a minimum of 70 cubic feet per man

^a Richards, R. H., Mining notes, vol. 2, 1905, p. 142.

per minute.^a When a person is sitting in repose as in a theater or meeting hall, 20 cubic feet of fresh air per minute is considered adequate provision by engineers making a specialty of ventilation, but much larger quantities are of course required when the men are working. The following table giving the results of a test conducted by Bernhardt Draeger,^b shows the amount of air breathed in the first minute after performing various kinds of work.

Quantity of air actually breathed in first minute after exertion.

Kind of work.	Subject A.	Subject B.	Subject C.	Average.
	<i>Liters.^a</i>	<i>Liters.^a</i>	<i>Liters.^a</i>	<i>Liters.^a</i>
Sitting 10 minutes.....	8.5	8.25	9	8.58
Walking 270 yards.....	10.5	11.3	11.7	11.2
Marching 550 yards.....	14.3	17.5	13	14.9
Running 270 yards.....	30	30	30	30
Rolling barrel weighing $\frac{1}{2}$ hundredweight.....	38	33	40.5	37.2
Running 550 yards.....	38	42	38	39
Race, 270 yards.....	b 52	c 61	e 59	d 51

^a 1 liter = 0.0353 cubic foot.

^b Time, 40 seconds.

^c Time, 42 seconds.

^d Time, 41 seconds.

These figures give the amount of pure air actually inhaled and exhaled, but, of course, in order that the products of respiration may be diluted sufficiently for the air in the confined space of a tunnel to be kept pure, a much larger quantity must be supplied. Assuming that 20 cubic feet is sufficient for a man at rest, and applying the ratio deduced from Draeger's table, it would appear that the following volumes of air (in cubic feet per minute) should be supplied for ventilation if the forms of exercise indicated were undertaken in a small room or in a tunnel: Sitting, 20; walking, 26; marching, 35; running, 70 to 90; rolling barrel, 85; racing, 130.

Although some members of the tunnel crew, such as the shovelers, ordinarily work as hard as men running or rolling a barrel, there are times when the work of the drillers more closely approximates the exertions required in walking; so, taking everything into consideration, it would seem that 75 cubic feet per minute per man should be adequate provision for tunnel ventilation. Assuming that a horse or mule requires two to three times the air needed for a person, 150 to 200 cubic feet per minute should be furnished each. At mine adits where any attempt is made for even moderate progress 8 to 15 men, and possibly 2 animals, are employed in or near the heading. Under these conditions 600 to 1,500 cubic feet of fresh air per minute would be required for purposes of respiration. It is true that some air is furnished by the exhaust from the drills, but

^a (Editorial), Carbon dioxide criterion for ventilation: Eng. and Min. Jour., vol. 90, Nov. 5, 1910, p. 899.

^b Glückauf, vol. 40, 1904.

their action is intermittent and the supply never adequate, so that much dependence can not be placed upon it; on the whole, it is much better simply to ignore this possible source when deciding upon the capacity of ventilating machinery.

Although the supply mentioned above is sufficient for ordinary requirements, a much greater, and indeed the maximum, demand for ventilation occurs immediately after blasting, when it is obviously important to remove the gas and smoke quickly so that the men may resume work with the least loss of time. The volume to be removed depends largely upon the amount of explosive employed; for customary charges under normal conditions it would probably not vary greatly from 60,000 cubic feet, a figure representing the average result of practical experience at tunnels where information bearing on this question was obtainable. It is true that ordinarily the air is seldom contaminated by the blast for more than 150 feet from the face. In a heading with a cross section of 70 square feet the blast would have a volume of only 10,500 cubic feet, and it might appear that the removal of this amount of bad air would clear the tunnel. Such might be the case provided the smoke would be removed instantly, but this result is, of course, not attainable in practice. The readiness with which gases become diffused must be taken into consideration, especially as it is customary immediately after blasting to turn a jet of highly compressed air into the heading. Such a practice is necessary because, to avoid injury from flying rock, the ventilating pipe rarely extends nearer the breast than 100 feet; so to remove the gases they must be forced out of the extreme end of the tunnel into the influence of the suction of the ventilating pipe. The result is that as a part of the bad air is removed its place is occupied by fresh air, which quickly becomes contaminated. It is therefore necessary to remove nearly six times the amount of foul air to clear the tunnel. In order to be considered good practice under ordinary conditions removal should take place in 15 minutes, requiring an exhauster capable of removing 4,000 cubic feet per minute.

This capacity, however, is necessary for only a few minutes after blasting. It is desirable, therefore, to have the fan or blower so arranged that it can exhaust for a short time at full load and then be run at a lower speed and supply the heading with the smaller volume needed for respiration. Such an exhauster was used at the Laramie-Poudre Tunnel. It was directly connected to a water wheel and ordinarily removed approximately 1,300 cubic feet by running at 100 revolutions per minute, but immediately after blasting the blower was speeded up to 300 revolutions per minute, when it ex-

hausted nearly 3,900 cubic feet per minute, clearing the heading usually in 15 to 20 minutes.

At the Rawley Tunnel an attempt was made to obtain the same result by operating the blower intermittently at or near full load. Although the operation of the blower or fan at full load for one-third of the time supplies the heading with an equal amount of air as when running at one-third capacity all the time, different results are obtained in practice. The purity of the air is not maintained so nearly constant with the intermittent system, and as the starting and stopping of the blower is usually dependent upon some man, its operation may be forgotten or neglected. This method of ventilating, therefore, can not be commended.

PRESSURE.

It is, of course, essential that the required amount of air be actually delivered to or removed from the heading. To do this pressure is necessary in order to overcome the frictional resistance to the flow of air in the pipe. This pressure must be generated by the fan or blower and may be either positive when forcing air in or negative when exhausting it. In either case the amount required depends upon the quantity of air passed and the size and length of pipe. Although the relations between these several factors are somewhat complicated, they are shown in the following formula advocated by G. S. Hicks, jr. :^a

$$q = 44.72 d^4 \sqrt{\frac{(P^2 - 14.7^2)}{lg}}$$

Where q = quantity of air in cubic feet per minute.

d = diameter of pipe in inches.

P = absolute initial pressure in pounds per square inch.

l = length of pipe in feet.

g = specific gravity of gas referred to air as unity.

p = required pressure in pounds per square inch.

From which, by transposing:

$$p = \sqrt{216.10 + \frac{q^2 l}{2000 d^4}} - 14.7$$

In which $p = P - 14.7$,

And $g = 1$.

It must be borne in mind that the formula is theoretical and does not take into consideration leakage, the extra friction due to elbows in the pipe, etc., but it is said to be based on good general practice

^a Engineering Tables, p. 12, P. H. & F. M. Roots catalogue 41.

for air and gas transmission and to give fairly satisfactory results. The table following, calculated from the formula, shows the pressure in pounds per square inch required to pass air through various sizes and lengths of pipe, assuming the quantity of air to be 4,000 cubic feet per minute—the value derived above as a suitable maximum capacity for a ventilating blower or fan.

TABLE 6.—*Loss of pressure when forcing 4,000 cubic feet of air per minute through various lengths and sizes of ventilating pipe.*

Diameter of pipe.	Length of pipe in feet.									
	1,000.	2,000.	3,000.	4,000.	5,000.	6,000.	8,000.	10,000.	12,000.	14,000.
Inches.	Lbs. per sq. in.	Lbs. per sq. in.	Lbs. per sq. in.	Lbs. per sq. in.	Lbs. per sq. in.	Lbs. per sq. in.	Lbs. per sq. in.	Lbs. per sq. in.	Lbs. per sq. in.	Lbs. per sq. in.
6.....	8.75	11.80	6.65	8.45	10.10	5.82	7.06	8.63	9.87	11.11
8.....	2.82	4.69	2.90	3.87	4.71	2.77	3.60	4.40	5.16	5.90
10.....	1.08	2.05	1.45	1.90	2.32	1.48	1.95	2.40	2.84	3.27
12.....	.50	.98	.76	1.02	1.25	.84	1.11	1.38	1.64	1.89
14.....	.28	.51	.43	.58	.70	.50	.67	.83	.99	1.15
16.....	.14	.29	.25	.34	.42					
18.....	.085	.17								
20.....										

If the pressure can not be increased to correspond with the length of pipe, the volume of air delivered is diminished, the size of the pipe remaining the same. This is illustrated in Table 7, in which a maximum pressure (P—14.7) of 1 pound per square inch is assumed.

TABLE 7.—*Maximum air capacities of pipes of different sizes and lengths when the initial pressure is 1 pound per square inch.*

Diameter of pipe.	Length of pipe in feet.									
	1,000	2,000	3,000	4,000	5,000	6,000	8,000	10,000	12,000	14,000
Inches.	Cu. ft. per min.	Cu. ft. per min.	Cu. ft. per min.	Cu. ft. per min.	Cu. ft. per min.	Cu. ft. per min.	Cu. ft. per min.	Cu. ft. per min.	Cu. ft. per min.	Cu. ft. per min.
6.....	685	485	815	705	630	575	780	710	660	610
8.....	1,410	1,000	1,425	1,235	1,105	1,005	870	780	710	660
10.....	2,465	1,745	2,245	1,945	1,740	1,590	1,375	1,230	1,125	1,040
12.....	3,890	2,750	3,300	2,860	2,580	2,335	2,020	1,810	1,650	1,530
14.....	5,720	4,045	4,610	3,990	3,570	3,260	2,825	2,525	2,305	2,135
16.....	7,985	5,645	6,190	5,390	4,795	4,375	3,790	3,390	3,095	2,865
18.....	10,720	7,580	8,055	6,975	6,240	5,695	4,930	4,410	4,025	3,730
20.....	13,950	9,865								

Table 8 shows the calculated pressure required to overcome frictional resistance in passing a volume of air equal to the rated capacity of the ventilating machine through pipes of the sizes adopted to convey the air to the headings of some of the tunnels visited in the field work when completed to their proposed length.

TABLE 8.—*Pressure required to force quantity of air equivalent to catalogue rating of ventilating machine to proposed length of tunnel through pipe chosen.*

Tunnel.	Rated capacity	Diameter of ventilating pipe.	Stated length of ventilating pipe when tunnel is completed.	Pressure required.
	<i>Cu. ft. per min.</i>	<i>Inches.</i>	<i>Feet.</i>	<i>Lbs. per sq. in.</i>
Carter.....	1,560	15	7,600	0.41
Central.....	5,540	19	9,500	1.93
Laramie-Poudre.....	3,900	14½	9,200	3.34
Los Angeles Aqueduct:				
Elizabeth Lake.....	6,350	18	13,000	4.14
Little Lake.....	2,500	12	3,000	1.23
Grapevine.....	2,500	12	1,500	.63
Lucania.....	3,120	18½	12,000	.87
Marshall-Russell.....	4,160	12½	11,000	8.30
Mission.....	2,500	10	13,000	10.25
Nisqually.....	2,400	14	5,000	.87
Rawley.....	2,500	12½	6,200	2.02
Roosevelt.....	4,800	16½	15,700	4.38
Siwatch.....	1,560	10	5,000	1.94
Snake Creek.....	4,650	16	14,000	4.27
Strawberry.....	4,000	14	19,000	7.50
Utah Metals.....	4,880	12	11,800	13.24

* This division contains a number of tunnels. The distance given is the maximum.

It will be observed in these examples that the pressures needed ordinarily range from 1 to 5 pounds, 2 pounds being roughly the average. At two of the tunnels in this list, in order to obtain the extra pressure required to furnish sufficient ventilation, it was necessary to use a "booster," as it is called—that is, to install a second blower some distance within the tunnel; by operating the two together the pressure otherwise attainable would be virtually doubled. At the Mission Tunnel, the "booster" was situated near the 5,500-foot station. At the Strawberry Tunnel, both machines had been placed in the tunnel at the time of examination, the first one at 4,000 feet and the second at 11,000 feet. Two other tunnels at the time visited had not penetrated far enough to require such additional equipment, but doubtless extra provision for obtaining pressure will become necessary with continued progress.

SIZE OF PIPE.

The necessity for high pressures, and hence the use of "boosters," may be obviated in large measure by the choice of ventilating pipe having diameters of sufficient size. The difference between a 12-inch and an 18-inch pipe often exerts a great influence on the ventilation of the heading, but even aside from added cost, indiscriminate enlargement is undesirable, every inch of space in the average tunnel being required for other purposes.

Transposition of the formula on page 85 gives—

$$d = \sqrt[5]{\frac{q^2 l g}{2,000(P^2 - 14.7^2)}}$$

which indicates the necessary diameter of the pipe in terms of the other variables. Table 9 following shows a number of solutions of this formula (assuming again that $q=4,000$ cubic feet of air per minute to be passed), and from it may be found the proper size of pipe for use with various pressures and distances.

TABLE 9.—*Diameter of pipe required in order to deliver 4,000 cubic feet of air per minute with different initial pressures and for various distances.*

Pressure.	Length of pipe in feet.									
	1,000	2,000	3,000	4,000	5,000	6,000	8,000	10,000	12,000	14,000
<i>Ounces per sq. in.</i>	<i>Inches.</i>	<i>Inches.</i>	<i>Inches.</i>	<i>Inches.</i>	<i>Inches.</i>	<i>Inches.</i>	<i>Inches.</i>	<i>Inches.</i>	<i>Inches.</i>	<i>Inches.</i>
1.....	21½	24½								
2.....	18½	21½	23	24½						
3.....	17	19½	21½	22½	23½	24½				
4.....	16½	18½	20	21½	22½	23	24½			
5.....	15½	17½	19½	20½	21½	22	23½	24½		
6.....	15	17	18½	19½	20½	21½	22½	23½	24½	
8.....	14	16½	17½	18½	19½	20	21½	22½	23	23½
10.....	13½	16½	16½	17½	18½	19½	20½	21½	22	22½
12.....	12½	14½	16	17	17½	18½	19½	20½	21½	21½
<i>Pounds per sq. in.</i>										
1.....	12½	14	15½	16	16½	17½	18½	19½	20	20½
1½.....	11½	13	14	14½	15½	16	17	17½	18½	19
2.....	10½	12½	13½	14	14½	15	16	16½	17½	17½
3.....	9½	11½	12	12½	13½	13½	14½	15½	15½	16½
4.....	9	10½	11½	12	12½	13	13½	14½	15	15½
5.....	8½	10	10½	11½	11½	12½	13	13½	14	14½
6.....	8½	9½	10½	11	11½	11½	12½	13	13½	14
8.....	7½	9	9½	10½	10½	11½	11½	12½	12½	13½

COMPARISON OF CENTRIFUGAL FANS AND BLOWERS.

Within certain limits the speed at which centrifugal fans are operated determines the volume of air delivered and the pressure generated, but the fans are incapable of producing pressures much greater than $1\frac{1}{2}$ pounds per square inch, and many of them are limited to 8, or even 5, ounces. Therefore as the frictional resistance against which air is to be forced or exhausted becomes greater through increasing lengths of pipe the pressure generated in the fan must be increased, by greater speed to the maximum limit at which the fan may be operated, and after that limit is passed the volume of air delivered necessarily becomes diminished with increased length of pipe. The positive blower, on the other hand, is capable of much higher pressures—8 pounds per square inch is easily attainable, and 15 pounds is possible with some makes—and in tunnel work, in which distances are as a rule great, the ability to deliver air against

high resistance is an important consideration in favor of the blower. It operates also at a much lower speed when delivering the same volume of air against an equal pressure (1:10 is considered a fair ratio), and this adjustment lessens the wear and tear upon belts and machinery. Because of its higher pressure, the blower makes it possible to choose a smaller diameter of pipe, a factor worthy of consideration, because not only the initial cost but also the space occupied must be taken into consideration. The first cost of the fan, on the other hand, is less than that of the blower, and to economize room and obviate the wear on the belt the fan may be connected directly to electric motors, the greater cost of low-speed motors tending to prevent this possibility with a blower.

SUMMARY.

In most cases a machine of the blower type, capable of high pressure, is better adapted for tunnel ventilation in which resistances are apt to be great. For the best results the ventilating pipe should be so arranged that the direction of the air current may be alternated at will, exhausting for a short time after shooting and blowing for the remainder of the time. The blower should be adjusted to operate at two capacities—a lower one, supplying 600 to 1,500 cubic feet per minute, as determined by the number of men and animals, and a higher one capable of exhausting approximately 4,000 cubic feet per minute. This adjustment would make it possible under ordinary conditions for the men to resume work in the heading about 15 minutes after shooting. The pressure generated in the blower must be properly adjusted to the size of the pipe and the length of the tunnel in order that the determined volume of air shall be actually delivered to or removed from the heading. The pipe chosen should be of such size that only a moderate pressure at the blower is required, due consideration being accorded such items as cost of pipe and the space such pipe must occupy.

INCIDENTAL SURFACE EQUIPMENT.

In connection with the blacksmith and repair shops mention should be made of the drill-sharpening machine and the compressed-air meter. The use of the former is quite common, being employed at a majority of the tunnels visited; but the latter, so far as could be learned, has been used only in one or two places, although there appears to be a field for its employment in tunnel plants.

DRILL-SHARPENING MACHINES.

Several types of drill-sharpening machines are used in the United States, each consisting essentially of a frame on which two cylinders are mounted, one vertically, the other horizontally, each containing a

reciprocating piston. Compressed air is employed as the motive power, the consumption, according to figures given by the manufacturers, ranging from 30 to 100 cubic feet per minute at a pressure of 85 to 100 pounds. Some device is necessary to hold the drill steel firmly in place. The sharpening is accomplished by means of suitable dies or dollies, which are either attached to or struck by the proper piston and forge the hot steel into the desired shape. The piston and die acting vertically are used for drawing out the corners of a broken or a very dull bit, or swaging out the grooves between the points, or insuring that the bit is of the required gage, whereas the horizontal die sharpens the cutting edges. With a suitable set of dies the machine may be used also for the construction of new bits from ordinary drill steel.

The use of a sharpening machine results in some saving of labor cost, for only one operator of medium skill is required. Such a man can ordinarily turn out several times the work of a skilled blacksmith and helper sharpening bits by hand. One manufacturer claims that his machine when handled by an expert is capable of sharpening 250 drills per hour, but he states also that half that number, under normal conditions, is satisfactory. With another type the capacity is given as 60 to 100 sharpened drills per hour. The lowest of these figures is more than ample for the usual requirements of tunnel work as, according to figures obtained at tunnels visited, the number of drills ordinarily sharpened ranges from 100 to 200 per day, although in hard ground the use of as many as 400 was reported.

The labor saved in the blacksmith shop is only a minor consideration, however, for the real superiority of the machine over hand sharpening lies in its ability to turn out perfect bits. As the progress in tunnel driving is often largely determined by the time required to drill a round of holes, this important part of the work deserves careful attention. It has been demonstrated repeatedly by practical experience that on comparing the cutting qualities of a machine bit with one sharpened by hand there is a marked difference in favor of the former. This is due to the fact that the bits come from the machine true to gage, thus greatly reducing the danger of binding or sticking in the hole; there is, therefore, less delay in drilling and a smaller loss of time from this cause for the driller and helper, or perhaps the entire crew, and there is less likelihood of "lost" holes. Then, too, the bits, being correctly shaped and properly sharpened, not only "stand up" better and stay sharp longer, but they also drill faster, and it is not necessary for the drill crew to change steel so often, thus reducing another source of delay. The use of drill-sharpening machines at the ordinary tunnel plant is, therefore, strongly recommended not only for its saving of time and labor both in the blacksmith shop and in the heading, but also for its ability to make

bits whose superior drilling qualities will easily pay a large return upon the money invested in the machine.

AIR METERS.

Air meters are of various types. In one of them the volume of air is measured by causing the air to impinge consecutively upon a number of turbine wheels mounted on a common shaft connected with a registering device by a master gear. The machine is calibrated to read in cubic feet per minute of free air and is claimed by the manufacturer to give accurate measurements of air under varying pressures. A second type operates upon the principle that, with a uniform difference of pressure on both sides of an orifice and a constant initial pressure and temperature, the quantity of air passed is proportional to the size of the orifice. In this machine the difference in pressure on the two sides of the diaphragm is kept uniform by the constant weight of a taper plug that closes the orifice until the difference in pressure is sufficient to raise the plug and support it. The taper is so designed that the amount of air passed through the orifice is directly proportional to the rise of the valve, and this movement is multiplied and transmitted to a needle which records it upon a moving sheet of paper, thus affording a means of measuring the volume of air passed. A third type involves a device for determining the pressure due to the velocity of the flow of air in a pipe, which is proportional to the amount of air passed if the temperature and initial pressure are constant, and for transmitting that pressure to one arm of a U tube filled with mercury. The tube is balanced on knife edges, and as the pressure causes a flow of mercury to the other arm, the balance is disturbed and the tube is deflected, the amount of deflection being commensurate with the pressure and hence with the flow of air. The deflection is transmitted by levers to a recording needle. In a fourth type, although only a proportional volume ranging from 0.125 to 8 per cent is actually measured, the recording device registers in terms of the full 100 per cent volume.

Any of these meters may be used to determine the quantity of compressed air delivered to a purchaser. Their most important use, as far as tunnel work is concerned, is in determining the quantity of air used by rock drills. It is well known that all pneumatic rock drills show an increased air consumption, due to leakage, etc., as the various parts become worn through use. This fact is quickly discovered in practice, and many actual tests bear out the statement that after the steady use of the ordinary rock drill for six months or a year this loss will range from 20 to 40 per cent. This additional air is not only expensive to compress but, what is of more importance, the efficiency of the drilling machine is lowered at the same time, and the man

operating it is unable to do as much effective work, thus entailing further loss. If the drill repairman has to guess at the air consumption, it is very difficult for him, even though he be an expert mechanic, to send from the repair shop back to the heading a drill that will do as good work as when it was new. But if the shop is provided with some means of determining the air required by the drill the repairman is much better able to remedy the defects and make the proper repairs. This results in a saving of expensive power and increases the efficiency of the drill and the amount of work done by the driller. Moreover, every time the drill leaves the repair shop it is very desirable to keep a record not only of the cost of its repair but also of its present air consumption, in order that upon its next return a comparison may be made with the last record as well as with the nominal air requirements. By such a course, if the air consumption is excessive necessary repairs may be made that would perhaps have been unsuspected otherwise, and at the same time the manager may keep an accurate statement of drill repairs and weed out inefficient drills. The following sample drill record ^a gives a rough outline of such a system:

DRILL RECORD.

Tool, *Piston drill.* Maker ----- Size, $2\frac{1}{4}$ inches.
Purchased, 2/1/10. Serial No. 123456. Shop No. 12.
Normal air consumption 90 cubic feet per minute, at 75-80 pounds.

Date.	Air consumption.	Pressure.	Repairs.
February 24, 1910	94	76	2 slide rods.
March 10, 1910.....	99	78	2 pawl springs.
			1 leather cup.
			1 chuck bolt.
			1 chuck key.
May 10, 1910.....	^b 128	75	1 air chest and valve. ^b
After repairs	96		2 piston rings.

^a By courtesy of the Excelsior Drill and Manufacturing Co.

^b Excessive air consumption corrected by repairs indicated.

UNDERGROUND EQUIPMENT FOR DRIVING TUNNELS.

Underground equipment for driving mine tunnels includes rock drills, cars for removing broken rock, and some means of hauling them to the surface, together with the necessary tools, track, pipe, hose, lamps or candles, and other incidentals. It may also include telephones or other means of communication with the portal and, if the tunnel is being driven down grade or from a shaft, pumps to remove water.

ROCK-DRILLING MACHINES.

TYPES.

As a rule rock-drilling machines are classified primarily according to the motive power by which they are operated. The great majority of those used in tunnels are of the pneumatic type, but hydraulic drills have been employed and electric drills are coming into favor. For surface work steam is sometimes substituted for compressed air by making a few minor alterations in pneumatic drills, and machines using gasoline power are also to be found on the market; but the difficulties of disposing of exhaust steam with the former and of the products of combustion with the latter prevent any extensive use of these types underground. Some of the principal features of the various rock drills employed in tunnel work are described below.

PNEUMATIC DRILLS.

The pneumatic rock drill consists essentially of a cylinder containing a piston or a hammer that is reciprocated by the proper admission, application, and release of compressed air. In the piston type of air drill, a drill steel provided with cutting edges is made alternately to strike and recede from the rock by the movement of the piston to which the steel is firmly attached. In the hammer drill (Pl. III, A) the steel does not reciprocate, but is held loosely against the rock to which it merely transmits blows received from a moving hammer. Piston drills are almost without exception mounted in a cradle that may be attached to some rigid support while the drill is in operation, but may be easily removed when necessary; a screw thread is provided also, permitting the drill to be fed forward in the cradle as the hole grows deeper. In some types of hammer drills, especially those used for stopping and trimming, the cradle is omitted, and the drill either is held in the hand or is provided with a tele-

scoping feed operated automatically by compressed air. In either type some device is required to rotate the drill steel in order that the cutting edges of the bit may not strike repeatedly in exactly the same place. In cradle-mounted drills this result is generally accomplished by a mechanism, consisting of rifle bar, ratchet, and pawls, that is arranged to turn the piston or hammer, this in turn rotating the chuck holding the drill. If the telescoping feed is employed the entire machine must be rotated by hand. Figure 9 shows a section through a piston pneumatic rock drill and gives a list of the principal parts.

Pneumatic drills are often differentiated by the method employed in controlling the admission of air to the cylinders. This may be accomplished by tappet, air-thrown, or auxiliary valves, or by direct

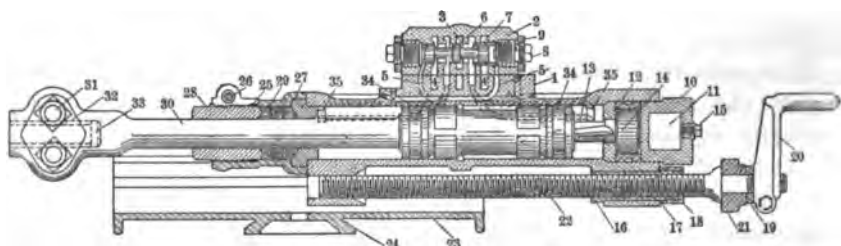
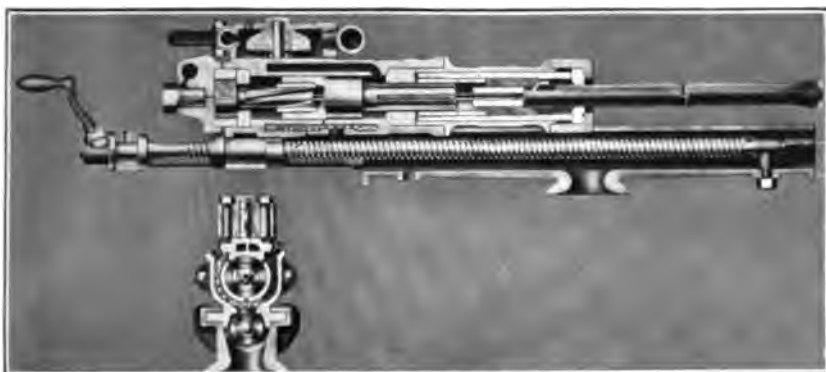


FIGURE 9.—Section of piston rock drill. 1, Cylinder; 2, steam chest; 3, inlet port; 4, exhaust ports; 5, valve; 6, valve bushing; 7, valve bushing; 8, buffer; 9, check nut; 10, top head; 11, oil chamber; 12, ratchet ring; 13, rifle bar; 14, ratchet; 15, plug; 16, feed nut; 17, lock washer; 18, check nut; 19, washer; 20, feed handle; 21, yoke; 22, feed screw; 23, shell; 24, trunnion; 25, lower head; 26, clamp bolt; 27, bushing; 28, gland; 29, packing; 30, piston; 31, clamp bolt; 32, chuck bushing; 33, chuck button; 34, piston rings; 35, cylinder ports.

regulation of the air supply by the movement of the piston or hammer itself.

The action of the tappet valve is indicated in Plate IV, 4, which shows a section through a drill equipped with such a valve. As in operation the piston moves from the position shown in the figure toward the lower end of the cylinder, the crank end of the tappet rises and the other end drops into the depression of the piston, thus producing around the tappet pin a slight rotation sufficient to move the slide valve. This movement admits air against the lower end of the piston, at the same time connecting the upper end of the cylinder with the exhaust pipe. The piston therefore starts in the other direction and a similar, but reverse, process takes place.

The operation of the air-thrown valve is somewhat more complicated than that of the tappet valve. In figure 9, showing a drill equipped with the usual form of air-thrown valve, the action of the valve may be traced as follows: In the figure the piston is indicated as just starting on the down stroke, the valve being so placed that live air is entering the top cylinder port 35 from the air inlet port 3 by way of the connecting passages indicated by dotted lines, while



A. SECTION OF IMPROVED TYPE OF HAMMER DRILL FOR TUNNELING.



B. TUNNEL CAR USED AT NISQUALLY TUNNEL.

at the same time the front of the cylinder is connected with the exhaust 4 by the lower cylinder port and its airways. The upper end of the "spool" of the valve is connected with the lower end of the cylinder, and hence with the exhaust, by the reverse port 5. As soon as the piston in its travel uncovers the other reverse port 5, shown by dotted lines, pressure from the upper end of the cylinder will be transmitted to the lower end of the spool and will throw it against the upper end of the valve chest, and this movement will alternate the connection of the ports for live air and exhaust, thus reversing the piston. A similar process is then repeated on the upstroke.

In a recent modification of the usual air-thrown valve the spool is replaced by a cylindrical shaft carrying two flat wings, which somewhat resemble those of a butterfly. The operation of this valve is indicated in figure 10. In the figure, upper view, the piston P is represented as about to start on the forward stroke. The valve is thrown so that live air is permitted to enter through the supply ports S, S2, and SS2, while the spent air in the front end of the cylinder is exhausting through the ports EE1, E1, and the exhaust E. As soon as the piston in its forward move-

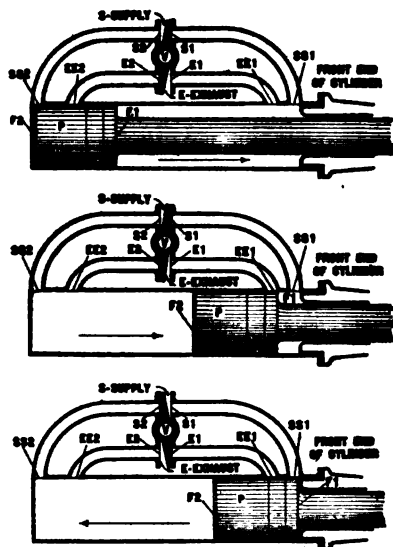


FIGURE 10.—Three views of butterfly valve, showing its action.

ment uncovers the exhaust port EE2, live air will pass through EE2 to E2 and its pressure on the valve at E2 will balance its pressure on the opposite wing of the valve facing port S2. The valve will then be in equilibrium, but will be held stationary, with the ports S2 and E1 open because of the impact of the air opposite S2. Near the end of the stroke, however, the piston closes the exhaust port EE1 and in passing from EE1 to F1 it compresses the air which is trapped in the clearance space at the end of the cylinder. This cushion pressure, communicated through the cylinder ports SS1 to S1, is sufficient to throw the balanced valve to the position shown in the middle view. Live air is then admitted through S1 and SS1, the exhaust ports EE2 are opened, and the piston starts on the return stroke.

One form of auxiliary valve used on a well-known piston drill is described as a mechanism in which the strains, shocks, and jars to

which the tappet or rocker is subjected are transferred from the main valve, with its vital and delicate functions, to a smaller auxiliary valve weighing only a few ounces, especially designed to withstand this service. This drill is illustrated in Plate IV, B.

When the drill is in operation one end of the auxiliary valve projects slightly into the cylinder and is thrown by the piston in its travel. The movement is perfectly free and very short—only enough to uncover a small port and release pressure from one end of the main valve, which is at once thrown by the resulting unbalanced pressure, opening wide the main port and admitting compressed air to the other end of the piston for the return stroke. The auxiliary valve is simply a trigger that releases the main valve.

In another form of auxiliary valve (fig. 11), the main air-thrown spool is controlled by two auxiliary valves consisting of steel balls that are positively actuated by the movements of the piston. In figure 11 the piston *a* is represented as having just started on the down stroke. Compressed air is entering the upper end of the

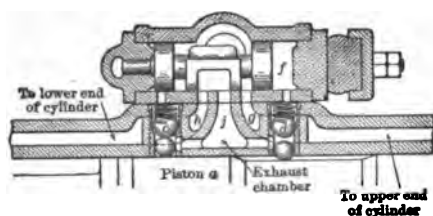
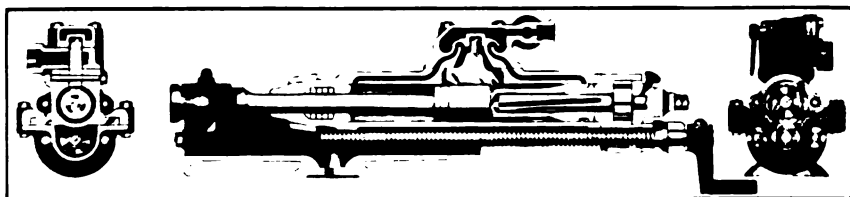


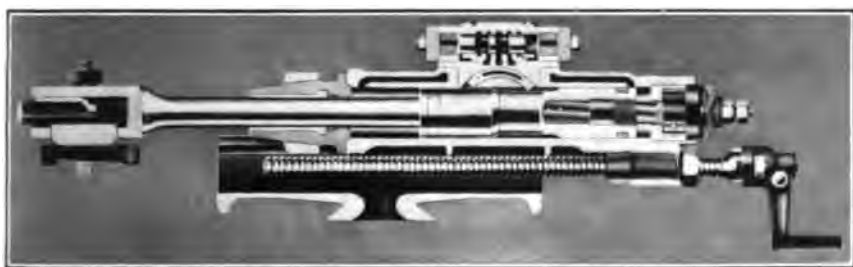
FIGURE 11.—Section of steel-ball auxiliary valve.

cylinder through the port *g* and the spent air in the lower end is escaping through the port *h* and the exhaust chamber *j*. At the end of the stroke the ball *c* will drop on its seat and the ball *d* will be raised, thus allowing the air in the end of the valve chest at *f* to exhaust past *d* through the port between the upper and lower balls. The unbalanced pressure thus produced throws the valve to the other end of the chest, which reverses the connections between the cylinder chambers and the inlet and exhaust ports. The piston therefore starts on the return stroke and a similar but reverse process takes place.

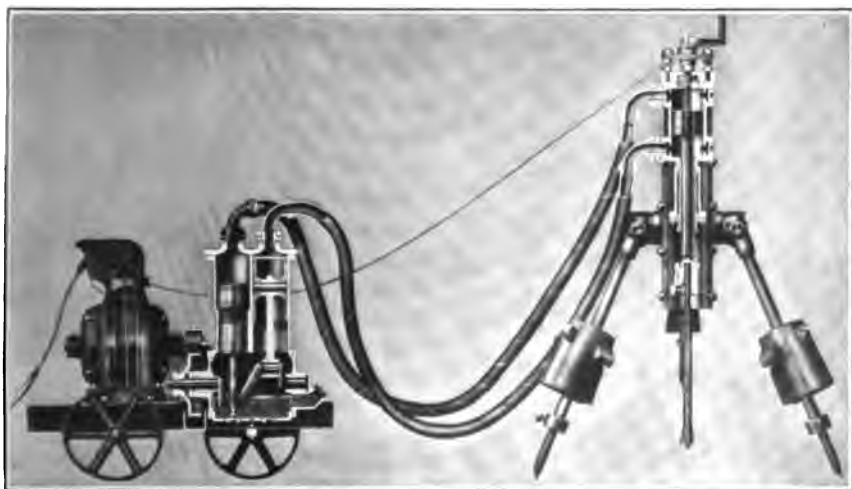
The valveless air-regulating mechanism, in which the movement of the piston itself covers and uncovers various ports, is employed almost exclusively on drills used for stopping only. Although rarely chosen for tunnel work, a brief description of this method of regulating air supply is warranted by its extensive use in its own field. The principle of operation is illustrated in figure 12, which shows cross sections through the cylinder of one make of valveless drill. In the position shown in the upper view of the figure air under pressure enters from the feed cylinder through the port *a* and passes to the front of the piston, where it exerts pressure at all times. The piston is forced back until the port *e* is uncovered, when compressed air passes through the port *f* and exerts pressure on the top of the pis-



A. SECTION OF A DRILL EQUIPPED WITH TAPPET VALVE.



B. SECTION OF TAPPET AUXILIARY-VALVE DRILL.



C. ELECTRICALLY DRIVEN AIR DRILL.

ton. As the area of this face is greater than that of the striking end, the piston starts forward. Live air is shut off when the port *c* is closed, but the piston is pressed forward by the expansion of the air until the exhaust port *b* is opened just as the blow is struck on the drill steel.

HYDRAULIC DRILLS.

The best-known hydraulic rock drill is, perhaps, one of the rotary type developed for use in the Simplon Tunnel, which consisted essentially of a hollow steel tube armed with teeth that were held firmly against the rock by hydraulic pressure while at the same time the tube was slowly revolved by a water-driven motor. This drill (fig. 13) is described by Prelini,* as follows:

This rotary motion is given by a twin-cylinder single-acting hydraulic motor, *e*, the two pistons, of 2½-inch stroke, acting reciprocally as valves. The cranks are fixed at an angle of 90° to each other on the shaft, which carries a worm,

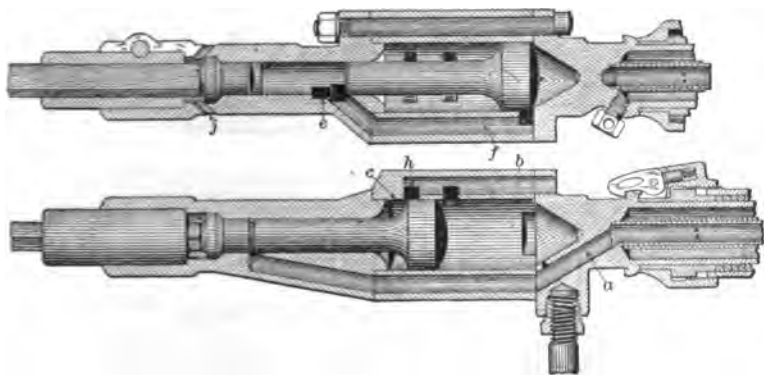


FIGURE 12.—Cross sections of valveless drill.

gearing with a worm wheel, *g*, mounted upon the shell *r* of the hollow ram *i*, and this shell in turn engages the ram by a long feather, leaving it free to slide axially to or from the face of the rock. The average speed of the motor is 150 to 200 revolutions per minute, the maximum speed being 300 revolutions per minute. * * * The pressure on the drill is exerted by a cylinder and hollow ram, *i*, which revolves about the differential pistons, which is fixed to the envelope holding the shell *r*. This envelope is rigidly connected to the bedplate of the motor, and, by means of the vertical hinge and pin *t*, is held by the clamp *v* embracing the rack bar. When water is admitted to the space in front of the differential piston the ram carrying the drilling tool is thrust forward, and when admitted to the annular space behind the piston the ram recedes, withdrawing the tool from the blast hole. The drill proper is a hollow tube of tough steel 2½ inches in external diameter, armed with three or four sharp and hardened teeth, and makes from 5 to 10 revolutions per minute, according to the nature of the rock. When the ram has reached the end of its stroke of 2 feet 2½ inches the tool is quickly withdrawn from the hole and un-

* Prelini, Charles, *Tunneling*, 1902, p. 103.

screwed from the ram; an extension rod is then screwed into the tool and into the ram, and the boring is continued, additional lengths being added as the tool grinds forward; each change of tool or rod takes about 15 seconds to 25 seconds to perform. The extension rods are forged-steel tubes fitted with four-threaded screws and having the same external diameter as the drill. They are made in standard lengths of 2 feet 8 inches, 1 foot 10 inches, and 11½ inches. The total weight of the drilling machine is 264 pounds and that of the rack bar when full of water is 308 pounds. The exhaust water from the two motor cylinders escapes through a tube in the center of the ram and along the bore of the extension rods and drill, thereby scouring away the débris and keeping the drill cool; any superfluous water finds an exit through a hose below the motors and thence away down the heading. The distributor, already mentioned, supplies each boring machine and the rack bar with hydraulic pressure from the mains,

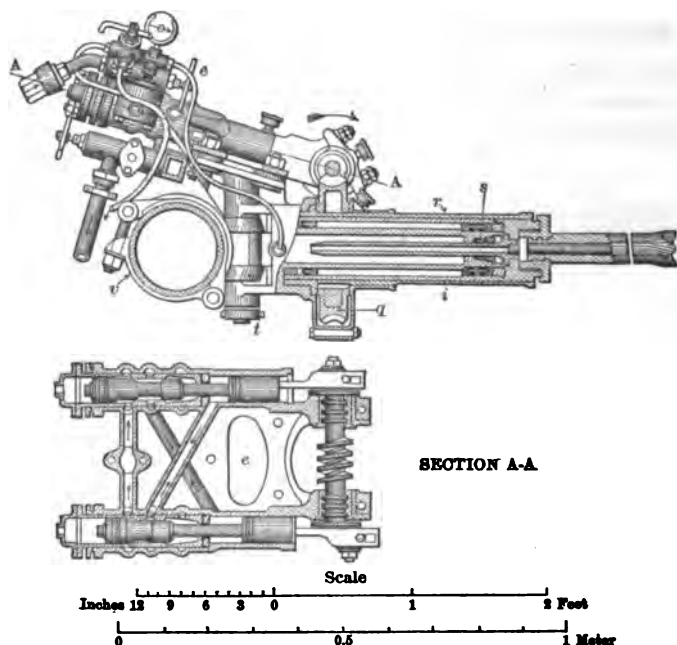


FIGURE 13.—Section of rotary hydraulic rock drill.

with which connection is effected by means of flexible or articulated pipe connections, allowing freedom in all directions. The area of the piston for advancing the tool is 15½ square inches, which under a pressure of 1,470 pounds per square inch gives a pressure of over 10 tons on the tool, while for withdrawing the tool 2½ tons is available.

A recently invented percussion hydraulic drill is described fully in the Engineer,* from which figure 14 and the following brief abstract are taken:

The drill consists essentially of a cylinder, in which is a piston, *c*, free to move, while at the other end of the cylinder is a flap valve, *d*, which is kept open by a spring. The interior of the cylinder is in communication with a

* New hydraulic rock-boring drill: The Engineer (London), Jan. 7, 1910, p. 24.

"striking tube," *fg*, at the end, *f*, of which is an air vessel. When the valve *h* is opened, water flows through the apparatus, out past the valve *d*, into the waste pipe *e*. The rush of water past the valves causes the pressure on the under side to be less than the pressure on the upper side, where the velocity is less. * * *

* * * When the velocity attains a certain value the difference of pressure is sufficient to close the valve, and the column of water in the striking tube is suddenly stopped. The kinetic energy of the water in the tube is communicated to the piston *c*, which is impelled forward with high velocity, and the drill which is at the end of it strikes a heavy blow on the stone or rock being bored.

The pressure in the interior of the cylinder is diminished by the moving out of the piston *c* * * * enough for the valve to open. Water then streams through the open valve. The piston is meanwhile being brought back to its original position by springs, but before it is right back * * *, the valve, *d*, closes, and the direction of motion is reversed by the hydraulic shock. The drill then strikes another blow as before. The actual apparatus is shown in

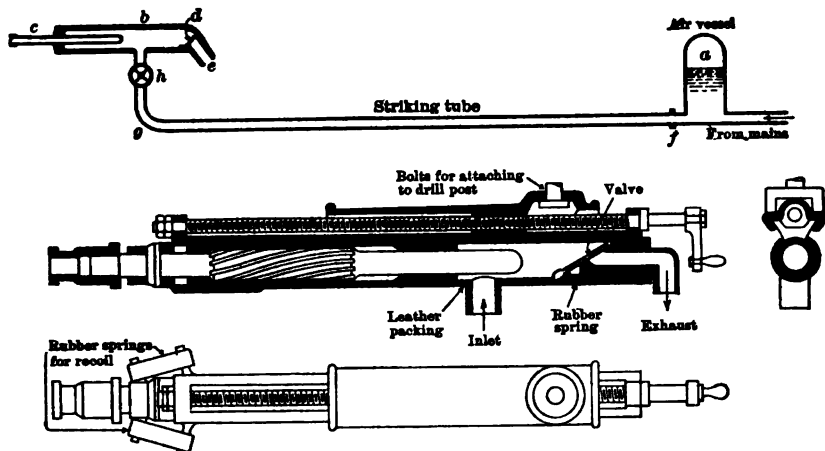


FIGURE 14.—Sections and connections of hydraulic percussion rock drill.

section and plan in figure (14 of this bulletin) which is roughly to scale, the over-all length being about 4 feet.

The actual magnitude of the blow depends primarily upon (1) the weight of the striking column; (2) the velocity of the water when the valve closes; and (3) the weight of the chisel and boring bar.

The velocity of the column is fixed by the velocity at the valve required to produce the necessary difference of pressure to close the valve—i. e., it is fixed by the stiffness of the spring controlling the valve. The rapidity of the blows is limited by the fact that after each blow the striking column is brought to rest, and it must be accelerated up to the requisite velocity before the valve will close. The rapidity of working depends, therefore, upon the pressure which is urging the column forward—i. e., it depends on the pressure in the supply mains. The actual magnitude of the blow is said to be unaffected by the varying pressure in the mains, and to depend only on the weight of the striking column and the strength of the spring controlling the valve. The inventor claims that machines of the type described strike from 20 to 30 blows per second, while the maximum speed of percussion machines of existing types is from 3 to 5 strokes per second.

One of these machines has recently undergone a series of tests at the Millbank pumping station of the London Hydraulic Power Co. The pressure used was 450 pounds per square inch. * * * The tests were carried out on a block of hard Portland stone. The diameter of the drill used was 2½ inches, and on an average progress was made in the stone at the rate of 10½ inches per minute. This is equivalent to the removal of 46 cubic inches of stone per minute. The drills stood up to the work so well that after holes aggregating about 25 feet in depth had been drilled it was not necessary to do anything to the edge. A stream of water plays on the chisel the whole time and serves the threefold purpose of keeping the chisel cool, of rinsing the bore hole, and of allaying the dust.

ELECTRIC DRILLS.

An electric rock drill consists primarily of an electric motor and a means of applying the power developed in it to the work of drilling rock. In some machines the motor is mounted directly upon the drill frame, but in others it is removed a short distance and connected to the drill by a flexible shaft or some similar device for transmitting power. Provision must also be made for preventing the shocks and jars developed by the impact of the drill steel upon the rock from being transmitted back to the motor, which is a machine incapable of operating for any length of time under such conditions. In many of the earlier models springs or cushions or some elastic material such as rubber were used for this purpose. These devices failed to give satisfaction either because of inability to do the work required or because of excessive wear, breakage, and annoyance. In two or three of the early models an ingenious attempt was made to avoid these troubles by taking advantage of the fact that if an electric current is passed through a spiral coil of wire a suitably placed bar of soft iron will be drawn into it. By providing two such coils or solenoids and causing the current to flow through them alternately an iron piston carrying a drill steel was made to reciprocate between them. In order to have the blow sufficiently smashing to be effective, however, a prohibitive weight of copper wire was needed for the solenoids. To-day practically all electric drills use compressed air in some manner to cushion the reaction of the blow. The compressed air possesses the very desirable characteristic of extreme elasticity and is not affected by wear and tear. In one machine, however, a hammer is made to strike the end of the drill steel by centrifugal force, the rebound giving the necessary flexibility.

One of the successful electrically driven rock drills that has been on the market for over five years is illustrated in Plate III, *C*. In this machine the drill piston is reciprocated by alternating pulsations of compressed air, created by a double-cylinder air compressor driven by a standard electric motor. Two short lengths of hose connect the air compressor to the drill, each running from one of the compressor cylinders to opposite ends of the drill cylinder. The air in the system,

which acts as an unwearing cushion between the pulsator and the drill, is never exhausted, but is simply used over and over. The drill is very simple—merely a cylinder containing a piston and rotating device—valves, chest, side rods, buffers, and springs being omitted, and the compressor has neither valves nor water jackets. The motor may be designed for either direct or alternating current as desired, and it is mounted with the compressor on a wheeled truck for easy handling.

A second air-cushioned electric drill of the piston type, but one in which the motor is mounted directly on the drill frame, is illustrated in figure 15. In this drill the motor *m*, which can be readily detached from the rest of the machine whenever the drill is moved, is connected by reducing gears to the crank shaft *s*, which drives the con-

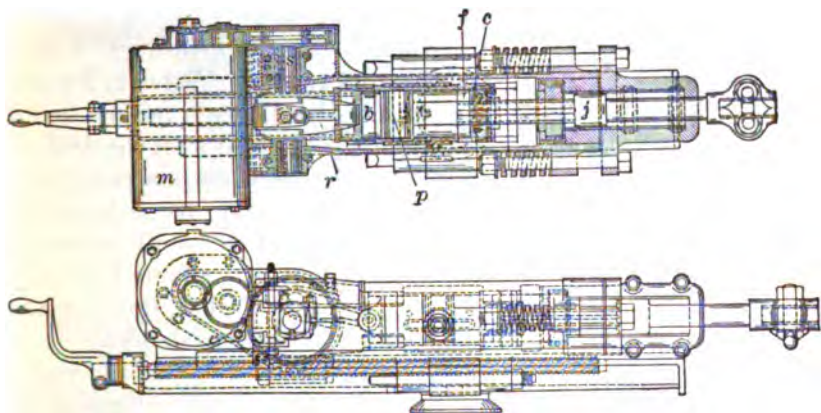


FIGURE 15.—Cross section of electric air-cushioned piston drill.

necting rod *r*. This is attached and gives a reciprocating motion to the cylinder *c*, which slides in suitable guides and contains a piston *p* provided with a chuck for holding a drill steel. As the cylinder moves forward air is compressed in the chamber *b* behind the piston and makes the piston move forward, which causes the drill bit to strike the rock. During the return stroke of the cylinder the compression of air in the other chamber, *f*, brings the piston back again with it. Rotation is obtained by means of standard spiral nut and ratchet. Details of the feed screw, the carriage, and other features are shown in the illustration.

In an electrically driven air-cushioned rock drill of the hammer type (fig. 16) power is transmitted by suitable gears and cranks from the motor to a piston and causes it to reciprocate in an air cylinder. The same cylinder contains at its other end a hammer which, however, is in no manner directly connected with the piston. As the latter starts on the down stroke it compresses the air in the space be-

tween it and the hammer, which is projected forward until it strikes the end of the drill steel. Just as it does so it releases the compressed air by uncovering an exhaust port controlled by a poppet valve. When the piston starts on the return stroke the exhaust valve

closes and the partial vacuum created pulls the hammer toward the piston. The latter in its travel uncovers an inlet port, also poppet controlled, admitting new air which destroys the vacuum. The momentum of the hammer would cause it to strike the piston, which again starts on the down stroke, were it not for the compression of the air entrapped by the closing of the poppet valve as soon as the vacuum is destroyed. The drill steel is rotated by the motor through a shaft, gearing, and ratchet. Water is forced to the cutting edge through hollow steel by a small pump supplied with the drill; but if water under pressure is already available the pump may be disconnected. Another feature of this drill is the automatic chuck which is adapted for using steel as it comes in the bar, thus obviating the necessity of forging shanks.

A fourth electric drill, also having an air-cushioned hammer, is illustrated in figure 17. In this drill, as the yoke *a* moves forward the piston *b* compresses the air in the chamber *c*, forcing the cylindrical hammer *d* against the anvil block *e*, which transmits the blow to the drill steel at *f*. On the return stroke of the piston the compression of air in the chamber *g* brings the hammer back in readiness for another blow. Water is forced through hollow steel by a small pump, whose plunger reciprocates with the drill piston.

So far as could be learned, the only electric drill in service to-day that does not use an air cushion is the one illustrated in Plate V, *A*. In the plate are shown two hammers which, although free to slide in their sockets in the revolving disk, are thrown out by centrifugal force and strike the anvil block which transmits the blows to the

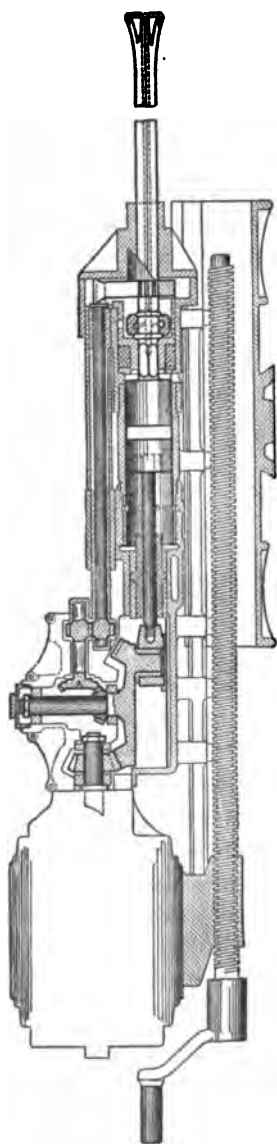


FIGURE 18.—Cross section of electric air-cushioned hammer drill.



A. ELECTRIC REVOLVING HAMMER DRILL, WITH MOTOR AND PART OF CASING REMOVED.



B. METHOD OF DUMPING ROCKER-DUMP TUNNEL CAR.

drill steel. The steel, which is held in a chuck rotated by a worm gear as indicated, is of the auger type, the spirals acting in the capacity of a conveyor for removing broken rock from the hole.

GASOLINE DRILLS.

Because the difficulty of disposing of the waste products of combustion, which are not only hot and disagreeable, but also contain gases injurious to the health of the workmen, makes the gasoline drill hardly suitable for service underground, and because, as far as could be learned, it has never been used in tunnel work, its design and construction will not be discussed here.*

MERITS OF EACH TYPE.

PNEUMATIC DRILLS.

The chief advantage of the pneumatic rock drill is its ability to withstand rough usage and still perform efficient service. The work

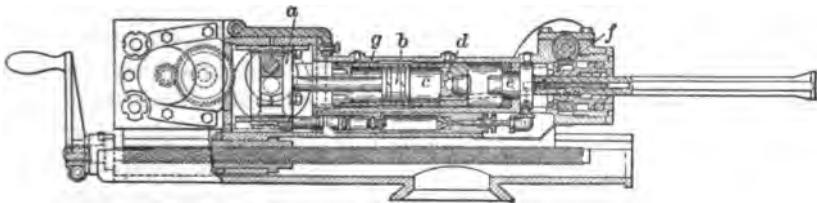


FIGURE 17.—Another type of electrically operated air-cushioned hammer drill.

of a rock drill is done necessarily under conditions that would quickly destroy almost any other type of machinery. It is subjected to constant and severe vibration when in operation, for although it is usually held firmly and securely, it can not be mounted rigidly. Lubrication, when supplied at all, is often administered in large doses most irregularly, and it is impossible to prevent sand and grit from getting into the machine, thus adding greatly to the wear and tear. In many cases men who operate it have no conception of its construction and ignorantly subject it to shocks and strains for which it was never designed, their first impulse when things go wrong being to seize a sledge hammer and hit the machine in the most convenient place. All drill runners of course do not belong to this type, but the description fits a much too large percentage of them. Everything considered, the rock drill, as some one has so aptly put it, must be

* For description of one of these machines having two explosion cylinders, see Eng. and Min. Jour., vol. 86, Nov. 21, 1908, p. 1008; Eng. News, vol. 60, Nov. 26, 1908, p. 575; and Min. and Sci. Press, vol. 97, Dec. 19, 1908, p. 852. For description of a drill, of English manufacture, in which a cam, driven by a gasoline engine, trips a spring-actuated piston, see Engineer (London), vol. 110, Sept. 30, 1910, pp. 365, 366, and Eng. News, vol. 64, Nov. 17, 1910, p. 538.

capable of being "cleaned up with a sledge hammer, wiped off with a scoop shovel, and still stay with you." This requirement necessitates the elimination of all unsuccessful features, the rejection of complicated parts that are not absolutely essential, the determination of the proper size and strength of those remaining, and the selection of materials having the proper stability and wearing qualities. It is only natural that the pneumatic drill, which has been undergoing development for more than 50 years, should be able better to cope with the conditions and to operate more steadily, with fewer interruptions and a lower cost for repairing broken or worn parts than any of the newer types.

Among other advantages of the pneumatic drill may be mentioned the fact that it furnishes some ventilation, that it does not require the introduction underground of electricity at comparatively high voltages (oftentimes a source of danger), nor does it require pipes strong enough to withstand the pressures needed for the rotary hydraulic drill. The air drill should not be relied upon too strongly for ventilation, however. In the first place, the supply of air is intermittent and is not forthcoming while the drill is stopped for the purpose of changing steel or moving it into position for a new hole, etc.; in the second place, the drills are not in operation immediately after the blast—the time ventilation is most needed—although it is true that the use of pneumatic drills makes it possible to direct a jet of compressed air into the heading at this time to assist in removing the smoke; and, finally, there are cases on record in which the exhaust from the drills not only did not deliver fresh air, but even filled the heading with carbon dioxide and other dangerous gases produced by combustion of oil and grease in the receiver, resulting, in one instance at least, fatally for several men. Again, at tunnels using electric haulage, the adoption of electric drills would simply add to a danger already present rather than introduce a new one, and in such cases the advantage of the air drill in this respect is not so important.

The most important disadvantage of the pneumatic drill, on the other hand, is its well-known lack of power economy. Since, as stated by Rix,^a "the tables set forth in the trades catalogues for the consumption of standard piston rock drills are fairly accurate," let us determine from them the power required for rock drills by using his estimate of 20 brake horsepower per 100 cubic feet of free air per minute. The lowest figure given for any type of rock drill used at the tunnels examined for this report is 65 cubic feet per minute at 100 pounds pressure, whereas drills using as much as 150 and even 175 cubic feet were very numerous. On this basis, then, without any

^a Rix, E. A., Compressed-air calculations: Address before the mining association of the University of California, Feb. 19, 1908.

allowance for loss of power through friction in the pipes or leakage in the machines when they become worn, pneumatic drills require the application of 13 to 35 brake horsepower at the compressor during the time the machine is operating. Although the rotary hydraulic drill employed in the Simplon Tunnel required as much as 13 horsepower^b (exactly the minimum figure just deduced for air drills) it is by comparing the power used in air drills with even the maximum of 6 horsepower for electric drills, many of which run on less than 2, that the large difference in power consumption is revealed.

As regards the different types of pneumatic drills used in tunneling, the piston machine has somewhat the advantage over the hammer type as regards reliability and efficiency in drilling holes vertically or nearly vertically downward. This reliability may be attributed, without doubt, to its simpler construction. It does not contain any mechanism for introducing a water spray through a hollow drill steel, it is not troubled by crystallization of metal parts from the repeated shocks of rapid blows, and it has a much greater range of feed. The last feature is important when the machine is handled by an inexperienced operator, giving as it does greater latitude before the piston begins to strike the front head. These considerations make the piston drill more nearly "fool-proof," and hence better adapted for use by ordinary drill runners—especially those in the Eastern States, who, as a rule, are neither as intelligent nor as careful as those in the West. Complexity of construction, however, should not be confused with the number of parts, for if this were taken as the standard and every screw, bolt, or nut counted separately it could be shown that the hammer drill is the simpler machine.

The greater efficiency of the machine in drilling holes that point downward was clearly brought out in the recent extensive drill competition in the Transvaal, according to the committee conducting the test, who reported that one of the main reasons for the better showing made by the piston drills underground was the fact that practically all of the holes drilled there were pointed downward. This is substantiated in several instances at tunnels in this country in which the excavation is accomplished by the heading-and-bench method; in such tunnels the piston drill is reported to have given better satisfaction in drilling the vertical holes required for the removal of the bench.

The hammer drill of the type used in tunnel driving, on the other hand, requires less power than the piston machine, is lighter, smaller, more easily handled in a restricted space, and is able to drill with greater speed holes that are horizontal, or nearly so, especially those pointing slightly upward such as are necessary under the ordinary

^b Comstock, C. W., *Great tunnels of the world*: Proc. Colo. Sci. Soc., vol. 8, 1907, p. 363.

methods in driving tunnel headings. In hammer drills the air consumption, and hence the power required, varies from 65 to 100 cubic feet per minute at 100 pounds pressure (catalogue rating) as compared with 135 to 175 cubic feet for piston drills. The weights of hammer drills range from 115 to 170 pounds, whereas the piston machines weigh from 280 to 400 pounds, and the dimensions of the former are approximately four-fifths those of the latter. The shorter length of the hammer machines makes it possible to start the cut holes nearer the side of the tunnel, thus obtaining a wider angle between each pair with a consequent increase in the chances of breaking the full length of the round of holes. The rate of drilling is of course largely dependent upon the character of rock penetrated, but the data of Table 10 below (in which it will be seen that piston drills, even in shale and sandstone, rarely drilled over 10 feet per hour, whereas the hammer drills in granite and other hard rock rarely fell below that figure, 15 and even 20 feet being not uncommon) seem to warrant the general statement that the hammer type has the greater speed in drilling the holes required in tunnel headings.

TABLE 10.—Comparative drilling speed of piston and rock drills as reported at various tunnels.*

Name of tunnel.	Type of drill.	Character of rock penetrated.	Drilling speeds per machine per hour.	Remarks.
Carter.....	Hammer.....	Granite.....	Feet. 10	Approximate.
Catskill Aqueduct: Roundcut.....	Piston.....	Shale.....	8	A fair average.
Wallkill.....	do.....	do.....	10.5	Normal conditions.
Central.....	Hammer.....	Gneiss.....	8-16	
Fort William (water).....	Piston.....	Trap.....	2	Phenomenally hard rock.
Gold Links.....	do.....	Gneiss.....	8-10	Approximate.
Joker.....	Hammer.....	Breccia.....	12-15	
Laramie-Poudre.....	do.....	Granite.....	15	Ordinary conditions.
Los Angeles Aqueduct: Little Lake.....	do.....	Hard granite.....	15.84	Average of 15 accurately timed shifts.
Grape Vine.....	do.....	Granite.....	10	Estimated average.
Lucania.....	do.....	do.....	13	Average of 3 drills.
Marshall-Russell.....	do.....	do.....	10-20	
Mission.....	do.....	Shale and sandstone.....	30	Medium ground.
Newhouse.....	do.....	Gneiss.....	10	
Nisqually: Headworks end.....	Piston.....	Rhyolite.....	8-10	
Discharge end.....	Hammer.....	do.....	10	Rock is much harder than at other end.
Ophella.....	Piston.....	Granite.....	8-10	
Rawley.....	Hammer.....	Andesite.....	15-20	Average of 4 accurately timed drill shifts, 19.7 feet.
Raymond.....	do.....	Granite.....	8-12	
Siwatch.....	do.....	do.....	10-15	
Snake Creek.....	Piston.....	Diabase.....	8-12	
Stillwell.....	do.....	Andesite.....	5-10	
Strawberry.....	do.....	Shale.....	13	Test run.
Utah Metals.....	do.....	Quartzite.....	6-8	

* Includes time used in setting up and tearing down column or bar, in shifting machine to new holes, and in changing steel, but does not include time used in mucking for set up or in loading, blasting, and clearing smoke.

^b During a competition test in which both drills were mounted on the same bar in order to obtain identical conditions of rock, etc., the piston machine drilled 21 feet per hour, whereas a hammer drill made only 20 feet. The conditions were unusual, however, because water under pressure was encountered in practically every hole drilled, and doubtless influenced the results greatly.

It is difficult to determine just how much of the greater speed of the hammer drill is due to the manner of attack, the water feature, the greater ease and speed in replacing a dull steel with a sharp one, or to the nonreciprocating drill steel, but there is little doubt that all these factors enter into the result. The piston machine when attacking the rock strikes comparatively slow, heavy, smashing blows that soon dull the cutting edges of the bit, especially if the rock be hard. Then until the steel is changed the penetration must be accomplished by crushing. Conversely, the more frequent blows of the hammer type, being lighter, do not dull the bit so quickly and penetration is effected by a chipping action which is speedier as well as more economical of power. The application of water through a hollow steel to the face of the drill hole, in addition to cooling the drill bit and preserving the temper of its cutting edges, affords a positive means of removing the cuttings promptly from the front of the bit. This not only prevents the recutting and grinding of material already broken, with a consequent saving of power, but it increases the efficiency of the machine because it enables the drill bit always to strike an uncushioned blow on "live" rock. Hammer drills having the water feature, however, are said to make a poor showing when drilling vertical holes. This is doubtless due to the fact that the velocity of the rising current of water in the drill hole is not sufficient to prevent the rock grains from settling against it to the bottom of the hole and interfering with the work of the drill.

The plunger action of the piston drills, on the other hand, although it is probably no more efficient in actually removing the rock grains, keeps them stirred up enough partly to obviate the difficulty. Any one who has experienced the trouble and delay of changing steels with the usual chuck in piston drills will appreciate the saving in time and energy resulting from the use of a chuck into which the drill needs only to be inserted. As in the hammer drill the steel does not reciprocate, the elimination of friction against the sides of the drill hole effects a considerable saving of power and prevents a retardation of the blow, even though, as has been argued, this advantage is partly offset by the loss of power in heating the hammer and drill end and in overcoming the inertia of the steel. An additional advantage of a nonreciprocating drill steel is the fact that it may be held against the rock at any desired point and a drill hole started wherever necessary without loss of time, a feature especially important where the face of rock is oblique to the drill.

Piston and hammer drills employed in tunneling are seemingly on an equal footing to-day as regards cost of drill repair parts, although until quite recently the piston drills had somewhat the advantage. From September, 1905, to March, 1906, hammer drills were

employed at the Gunnison Tunnel with a drill-repair cost per machine of 13 cents per foot of hole drilled; but when piston drills were substituted the repairs were reduced to 3 cents per foot.* Two years later (September, 1907, to August, 1908), in driving the last 3,000 feet of the Yak Tunnel, the cost of materials only for repairs to the hammer drills employed was only $1\frac{1}{2}$ cents, approximately, per foot of hole. At the Marshall-Russell Tunnel where hammer drills were employed, the average cost of drill repairs from June, 1908, to June, 1911, was only $1\frac{1}{2}$ cents per foot drilled. Piston machines were used at the Strawberry Tunnel from January, 1909, to the time of the authors' visit in September, 1911, the cost for repairs being nearly 24 cents per foot drilled. On the Little Lake division of the Los Angeles Aqueduct, where hammer drills were employed, the average cost of drill-repair materials from July, 1909, to May, 1911, as shown by the following record was only 24 cents per foot of tunnel excavated. As each of the two machines in the heading drills approximately 8 feet of hole for every foot of tunnel excavated, the cost per machine per foot of hole is $1\frac{1}{2}$ cents.

Cost of repairs for hammer air drills, Little Lake division, Los Angeles Aqueduct, July 1909, to May, 1911.

Name of tunnel.	Distance excavated.	Total cost of drill repairs.	Cost of drill repairs per foot of tunnel.
	<i>Linear feet.</i>		
1B, south.....	1,030	\$180.50	\$0.156
2, north.....	926	180.72	.195
2, south.....	419	64.75	.154
2A, north.....	460	46.28	.100
2A, south.....	375	55.50	.148
3, north.....	864	113.60	.131
3, south.....	2,149	505.01	.235
4, north.....	448	67.03	.149
4, south.....	725	215.48	.297
7, north.....	1,911	399.70	.209
7, south.....	1,024	493.46	.482
8, north.....	225	146.56	.651
8, south.....	1,334	530.52	.398
9, north.....	777	230.51	.297
9, south.....	2,479	404.94	.163
10, north.....	2,626	585.78	.223
10, south.....	1,776	577.24	.325
10A, north.....	1,373	303.06	.221
10A, south.....	1,756	359.27	.204
Average.....			.21

For 1910 and the first half of 1911 the repair cost of hammer drills at the Carter Tunnel was 2 cents per foot drilled. At the Lucania Tunnel the repairs cost one-half cent per foot drilled, but the hammer drills had been in use only one month at the time the tunnel was visited. The hammer drills at the Rawley Tunnel were new also,

* In addition to the cost of materials these figures include also a charge for the labor of the machinist making the repairs, which is not embraced in any of the values which follow. This fact must be considered in making comparisons.

the repairs for June and July, 1911, averaging 1 cent per foot of hole. These figures, which are based upon estimates furnished by managers or others in charge at the various tunnels, do not pretend to other than approximate accuracy, but they give a basis of comparison such as has been hitherto unattainable, although in making such comparisons the type of rock must of course be duly considered.

In spite of the development of other types of valve mechanism for air drills, the tappet valve, which was one of the pioneers in the field, possesses advantages that still keep it in demand for use on piston drills intended for certain kinds of work. As it is unaffected by condensed moisture, which greatly interferes with the action of some other types, it is especially adapted for use with steam or with air containing a large quantity of water vapor. Its distinctive advantage, however, is that its movement is positive—if the piston makes a stroke the valve must be thrown—hence there is no uncertainty or “fluttering” in the action of the drill.

The tappet drill is at a disadvantage when working in ground that will not permit the use of a full stroke, because the piston must travel far enough to throw the valve and hence too short a stroke is not permissible. Then, too, because some air will necessarily be trapped in front of the piston and compressed after the valve is thrown, the drill strikes a cushioned blow. This is not always a disadvantage; in elastic and “springy” rock an uncushioned blow will not give the best cutting effect, whereas in sticky material compression assists the piston in starting on the return stroke. The tappet is subject to strains and wear, which necessitate specially hardened material, not only in the tappet itself but in the bearing surfaces of the piston.

Under conditions that require a snappy, vicious blow with high air pressure, the ordinary air-thrown valve gives the best results. This valve is particularly applicable to hammer drills in which, because of the small size and weight of the hammer, it is essential that there shall be no cushioning of the blow, and the air-thrown valve is customarily employed on such of these machines as are not of the valveless type. When used with piston drills the air-thrown valve permits a variable stroke; it renders possible a change at will in the length of piston travel and the force of blow. The short stroke and light blow possible with this type of drill make it adapted to starting a hole or to drilling through seamy rock. After the hole is under way, or if the rock is solid, a full stroke is used to get the best efficiency from the machine. The air-thrown type of valve is not positive in its action, however, and is apt to be somewhat sluggish with air or steam containing much water. The claim is made for the butterfly type that it avoids this difficulty as well as most of the troubles caused by freezing and that it has a positive and at

the same time a flexible action which permits much higher speed than other valves.

The auxiliary valve is designed to combine the advantages of the tappet and the air-thrown valves while avoiding their defects. The lightness of the tappet auxiliary is said to prevent the injury or retardation of the piston and also to obviate the rapid wear of rings, piston, and cylinders caused by crowding against the opposite cylinder wall, due to an unbalanced tappet not readily moved. A drill equipped with this type of valve has a wide variation of stroke and delivers an uncushioned blow. The main advantage of the steel-ball auxiliary valve is its great resistance to wear and the cheapness of replacing its wearing parts. The claim is made for this valve that it assures a positive action of the drill without sticking or fluttering and yet possesses the necessary flexibility.

The valveless regulating of admission and exhaust has the advantage of simplicity and lighter weight owing to the elimination of the valve and valve chest. It also uses air expansively, which should result in economy of power. It entails a cushioned blow, however, thus reducing the drilling power where the rock is hard and tough; but for medium hard rocks, especially with high air pressure, the difference is said to be less pronounced because the lighter and more rapid blows chip rather than pulverize the rock and enable the drill to penetrate readily. One real disadvantage is the fact that as the cylinder becomes worn there is a leakage of air past the piston, increasing the air consumption and interfering with the accurate working of the drill.

HYDRAULIC DRILLS.

Among the advantages and disadvantages of the rotary hydraulic drill used at the Simplón Tunnel should be mentioned the fact that the power was delivered to the cutting edge without the shocks, jars, and strains due to percussion, thus eliminating one source of wear and tear. The machine also utilized a very high percentage of the power stored in the motive fluid, its efficiency being given by one authority as 70 per cent. Again, by passing a part of the waste water down the boring tube, chips and débris were promptly removed from the cutting edge, thus insuring the maximum boring power. On the other hand, the pressure required for operating this drill was enormous, ranging from 450 to 1,200 pounds per square inch, according to one writer, and according to another, 1,470 pounds. In any case the piping necessary to transmit the water under such high pressures must have been expensive to install and maintain. To withstand the back pressure, the drill also required extremely heavy and rigid mountings, which made it cumbersome and hard to move, so that it could not be easily placed for a new hole.

The percussion type of hydraulic rock drill can not as yet be said to have been demonstrated to be a practical success. It is an interesting possibility, however, because, like the hydraulic ram, it utilizes the shock that occurs in pipes at every stoppage of a moving column of water.

ELECTRIC DRILLS.

Among the advantages claimed for an electric drill of the pulsator type are saving of power, rapid drilling speed, simpler construction, and less trouble with fitcher holes. The motors used to operate the pulsator require 3 to 5 horsepower, a very small amount compared with the necessities of the ordinary pneumatic drill. Although it is true that the cost of power used by a drill is not the only item that determines its efficiency, such a marked difference in power consumption is an important consideration. Moreover, the pulsator electric drill has the full drilling speed of any corresponding standard air rock drill and has practically the same cost for wages and fixed charges. The pulsator type also eliminates many parts, such as valves, springs, and side rods, which are sources of trouble and unreliability in other rock drills. It is able also to strike a heavy blow because the pressure of air back of the piston is greatest just at the time of impact; and should the drill steel become caught in the hole from any cause, the machine does not cease running, as is the case with air drills, but the pulsator continues to exert several hundred alternate pulls and pushes on the drill steel per minute, which in most instances are sufficient to loosen the drill at once.

On the other hand, the combined drill and pulsator are cumbersome and occupy a large space, every inch of which is precious in the tunnel heading—a disadvantage that increases directly with the number of drills needed for the work. For tunnel work it is necessary either to expend extra time and energy in placing the truck and pulsator upon the muck pile, where they are subject to breakage if the muck is being removed simultaneously with the drilling; or to wait until the tunnel is cleared of débris before starting to drill, a procedure that is impracticable if speed in driving is required. But if there is no particular haste or if drilling and mucking are alternated, this disadvantage is not so serious.

The piston electric drill described on page 101 does away with the need of a pulsator, truck, and connecting hose, thus making a compact machine and one more comparable with an air drill. It is, however, rather heavy (weighing 490 pounds with the motor attached and 350 pounds without it) and is somewhat difficult to handle and move in a small heading. It has a marked advantage over air drills in power economy, operating as it does on 4 horsepower, and actual results show that its drilling speed is fully up to that of standard

piston pneumatic drills. At the Elmsford Tunnel of the Catskill Aqueduct these drills are reported to have attained a speed of 100 feet in 6 to 8 hours when drilling in a comparatively soft mica schist, but in the harder Fordham gneiss of the City Tunnel the rate was only 60 feet per shift of 8 hours. This drill is still in the process of development; the small defects that always appear in any newly designed machine when put to actual use need to be corrected, but the results attained with it on one part of the City Tunnel were very encouraging. One of the machines is reported to have operated there for more than five weeks, drilling over 4,000 feet of holes with only minor breakages, such as pawl springs.

The weight of the air-cushioned hammer drill and motor described on pages 101-102 is about 150 pounds less than that of an electric piston drill and motor. With the motor removed, although it weighs more than a pneumatic hammer drill, it is little heavier than a piston air drill of corresponding capacity. Its power consumption is rated at $2\frac{1}{2}$ horsepower, and in the tests on the Catskill Aqueduct 6 to 8 feet per hour was the average drilling speed attained in ordinary work, including delays. This speed was undoubtedly increased as the delays from breakdowns became less frequent. The drill was still in the process of being perfected at the time of examination, so no data could be obtained as to its reliability.

The other air-cushioned hammer drill (see p. 102) has been employed in several mines in Colorado, where it is reported to be performing creditable service.

The average power consumption of the rotary hammer drills (pp. 102-103) is about 1 kilowatt per hour ($1\frac{1}{2}$ horsepower). They were employed on the Elmsford contract of the Catskill Aqueduct and were reported as particularly efficient in comparatively soft rock, drilling at times as high as 100 feet per machine in an 8-hour shift.

CHOICE OF DRILL.

The factors to be considered in the selection of a rock drill for tunnel work are, on the one hand, the cost of power, attendance, maintenance, and fixed charges, and on the other the rate of drilling. the best drill being the one that combines all these factors in such a way as to develop the greatest drilling speed for the least cost. The power cost should embrace not only the actual power at the tunnel plant, including charges for labor, fuel, interest, and depreciation, but all losses in generation, in transmission, and utilization in the drill. The wages of the drill runners and all helpers required are just as much an item of operating cost as the charge for power. The cost for maintenance includes the cost of repair parts for the drill and the charge for the time of the machinist, together with the cost

of sharpening drill steel. The fixed charges should include interest and depreciation on the cost of the drills and a proportion of the administrative expenses. The rate of drilling, on the other hand, should not be based upon the speed of penetration while the drill is actually hitting the rock, but should include all delays caused by the drill, such as loss of time in preparing the set up, in shifting the drill into position for new holes, in changing drill steels, and any other interruptions properly chargeable against the machine.

Applying the specifications mentioned to the various rock-drilling machines, the hammer pneumatic drill is seemingly the one best adapted for use under ordinary conditions in driving mine adits and tunnels. Its power consumption is more than that of electric drills, but is about equal to that of the hydraulic and is less than that of the piston air drill. In the matter of attendance it has somewhat the advantage. Both the piston air and the electric types usually require at least two men—a drill runner and a helper—to operate each drill, and the hydraulic machine requires five men.* With the hammer drill a runner is necessary, of course, but one helper often is able to attend to two drills, or two helpers to three machines. We have just seen that there is practically no difference between the piston and hammer air drills and some of the electric drills as to repair cost. The multiplicity of parts in the rotary hydraulic machine, however, is said to have been a source of much trouble in this respect. Theoretically the hammer drills do not dull the steel so rapidly, and hence should call for less of the blacksmith's time for sharpening bits. Practically this difference is not important, because under ordinary conditions the blacksmith is rarely overtaxed, and hence the extra labor of sharpening a few bits more or less is not noticeable on the cost report. The fixed charges are such a small part of the total cost of drilling that any discrepancy in them is rarely, if ever, large enough actually to decide the question. The rate of drilling is really the greatest factor in favor of the hammer type ordinarily used in tunnels. Not only does it penetrate faster when actually drilling but, as it has not the excessive back pressure of the piston machine, it can be employed with a lighter set up, with a consequent saving of time. Then, too, its ability to start a hole at any desired point and to drill rapidly holes that point upward enable it to be used advantageously on a horizontal bar, with a saving of the one-half to one and one-half hours required to remove the débris before setting up the vertical column generally used in tunnel headings for piston air drills. The hammer drill not only demands less time for changing drill steels but conserves energy as well.

* Pretini, Charles, Tunneling, 1902, p. 105.

If large tunnels are being excavated by the heading and bench method and if a large number of holes are being drilled downward, or if, because of acidulous mine water or some other reason, the water feature of the hammer drill would be unsatisfactory, the piston pneumatic drill would doubtless give equally if not more satisfactory results. Or if speed is not especially required and the drilling and mucking shifts can be alternated, the pulsator electric drill, with its large power economy, might prove the most efficient. Moreover, if the self-contained electric drills continue to be improved as they have been recently, their greater economy of power will without doubt soon outweigh their lower drilling speed and present higher maintenance charges, especially at places where electricity is readily available. On this account their development should be closely watched.

HAULAGE.

TUNNEL CARS.

Most students of tunneling methods concede that an essential, and possibly the chief, feature of the problem is the rapid removal of débris produced in blasting; but it is commonly not so well recognized that the speed with which this may be accomplished is greatly influenced by the size of the tunnel car. Large cars even when empty are heavy and cumbersome, but when full of rock they can be handled only with the greatest difficulty. To remove such a car from the heading and replace it with an empty one requires either several extra men to assist in the work, or a horse or mule must be provided for the purpose. In addition to being unwieldy, large cars occupy a greater proportion of the actual space in the heading, constricted enough at best, thus preventing the shovelers from working to the best advantage; the added height involves a waste of energy because each shovelful of rock must be lifted a greater distance, making it impossible for the men to handle as much material in a given time. With large cars it is necessary to maintain a switch or siding near the end of the tunnel in order to permit the empty cars to pass the loaded ones, and time and labor must be expended frequently in relocating the switch nearer the heading to keep pace with the tunnel advance. The smaller car, on the other hand, when empty can be tipped off to one side out of the way and replaced easily when needed, thus giving a clear track for a loaded car and obviating the necessity for a switch. In case of derailment, by no means rare in practice because of the poor condition of most tunnel tracks, the large car, even when empty, is harder to replace, and when full must sometimes be entirely unloaded before it can be put back on the track. Although a larger number of the smaller cars, each

of which occasions some delay in its arrival and departure, is necessary to remove the same amount of *débris*, the authors' observations, based upon a study of actual conditions at a large number of tunnels, are that with proper organization greater progress is attainable by using smaller cars, preferably having 15 to 25 cubic feet capacity. The tendency at many American tunnels is toward the use of much larger cars, especially where electric haulage is employed; but the use of large cars, when analyzed, has been shown to be a handicap rather than a contribution even in those tunnels in which creditable progress has been made.

In design the cars at a majority of the tunnels visited follow the standard mining types with tilting bodies, but at a few of them other types were employed to meet special conditions. A car with a side-dumping, tilting-box body used in the west end of the Gunnison Tunnel is illustrated in figure 18. End-dumping cars are similar to this, except in that the hinge is transverse instead of longitudinal and the door is situated at the end instead of the side. The car used at the Laramie-Poudre Tunnel, which is shown in figures 19 and 20, was of the turntable type, which permitted dumping from both sides of the track as well as between the rails. As the system of car handling in the headings at this tunnel necessitated throwing all of the cars over on their sides once, and nine-tenths of them twice, on each trip, the connections between the trucks and bodies of the cars were made unusually strong. The turntables were fitted with two concentric rings (fig. 20), and the locking mechanism for securing the bodies to the trucks was so designed that when the releasing lever was fastened in place the cars were as rigid as if the bodies were riveted to the axles. A car of the rocker type (fig. 21 and Pl. III, *B*) was used with very satisfactory results in the tunnels of the Los Angeles Aqueduct. At the Nisqually Tunnel a similar car (Pl. III, *B*) but with a slightly different locking device was employed. In order to obviate the tilting body, the car at the Utah Metals Tunnel was constructed with the floor permanently inclined

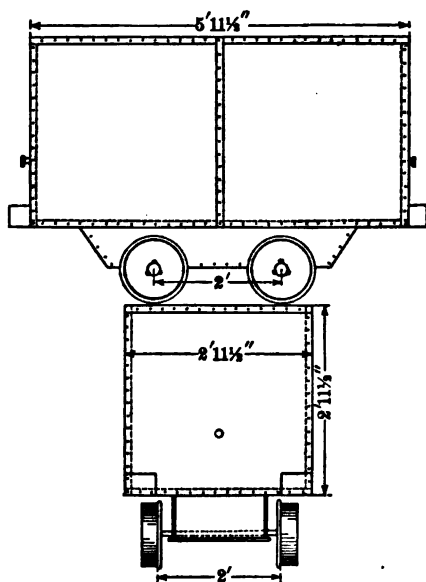


FIGURE 18.—Tunnel car used in the west end of the Gunnison Tunnel, *side and end views.

toward the side door; at the Carter Tunnel the car used was of the gable type, in which the floor slopes away from the center toward doors on each side of the body. At the east end of the Gunnison

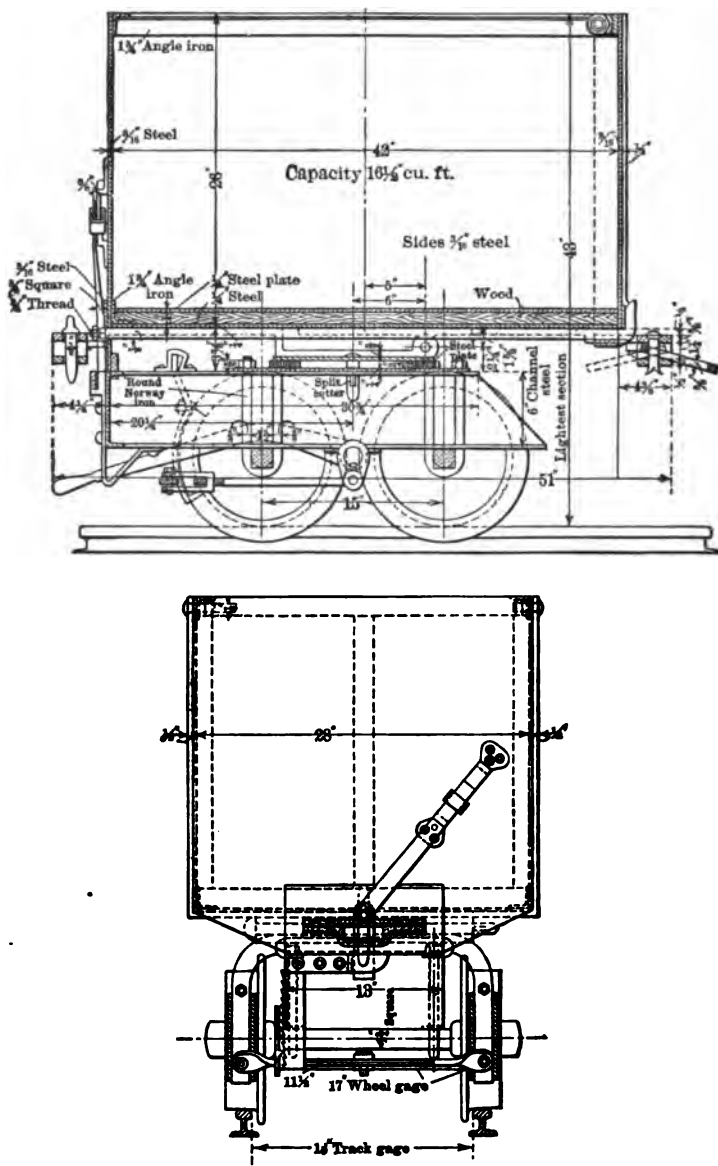


FIGURE 19.—Tunnel car used in Laramie-Poudre Tunnel, side and end views.

Tunnel a simple open box car with the body bolted directly to the truck was employed, and similar cars are now in use in the Strawberry and Newhouse Tunnels. Although this car is ideal as regards

its simplicity, it requires special equipment for dumping, because the entire car must be turned completely over. The following table

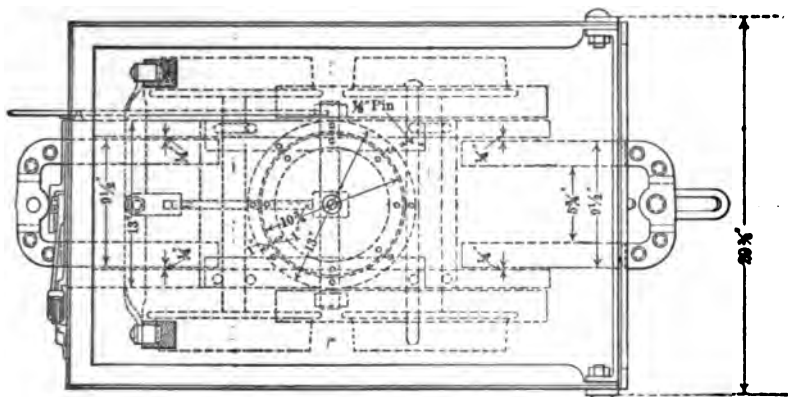


FIGURE 20.—Tunnel car used in Laramie-Poudre Tunnel, plan.

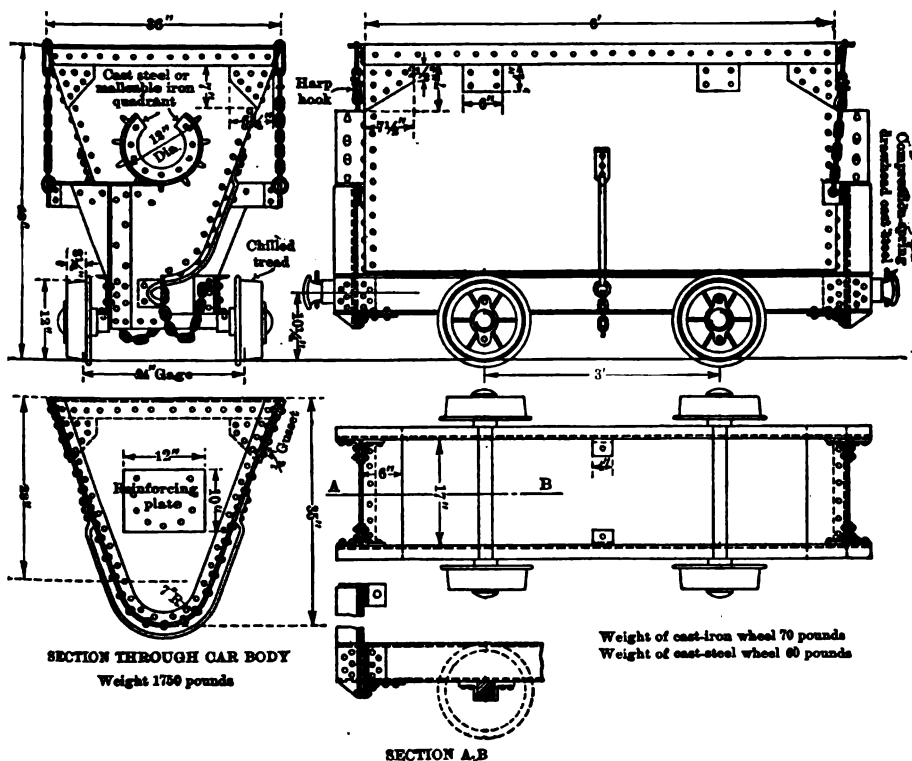


FIGURE 21.—Rocker-dump tunnel car used on Los Angeles Aqueduct.

contains suggestive data concerning the cars used in tunnels and adits in the United States:

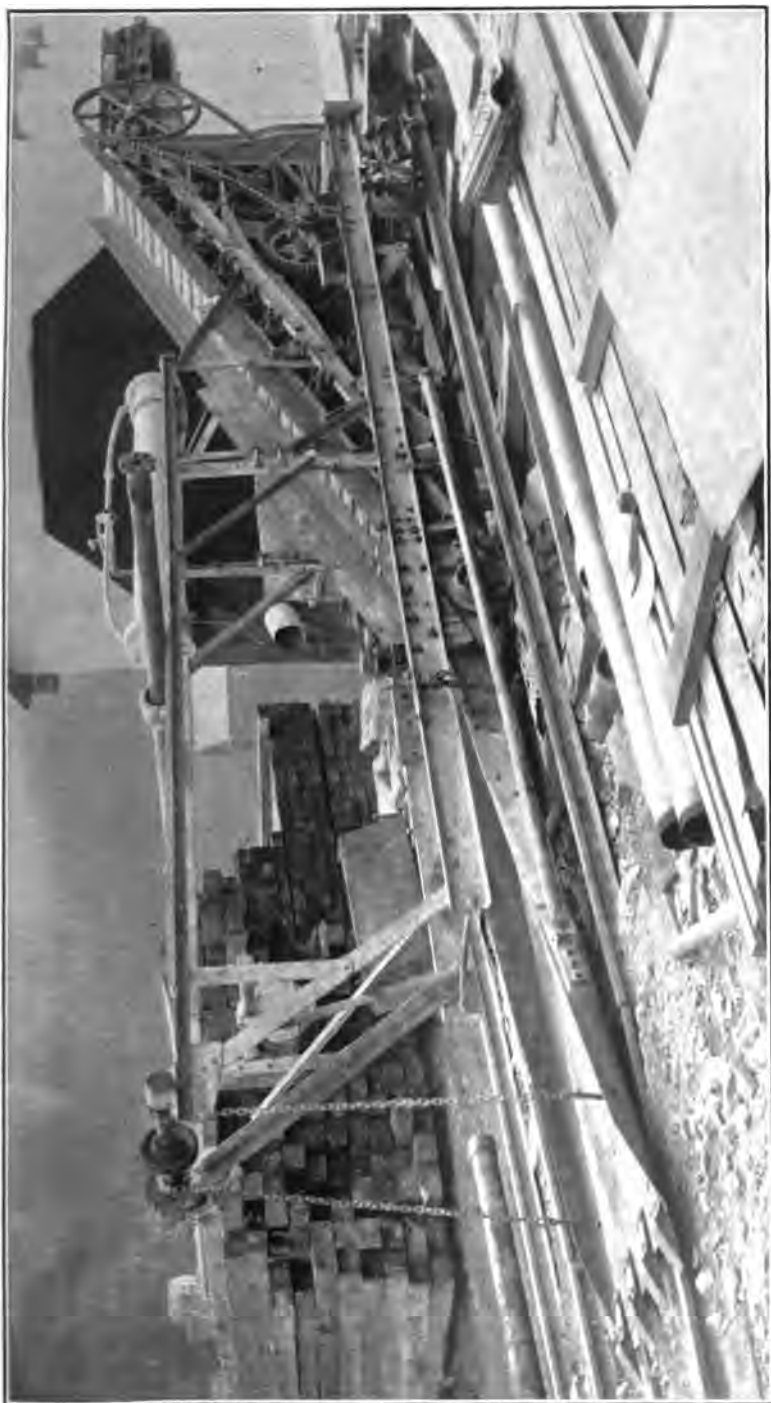
TABLE 11.—Data concerning tunnel cars used in the United States.

Name of tunnel.	Type of car.	Capacity.	Track gage.	Weight of rail.	Means of haulage.	Remarks.
		<i>Cubic feet.</i>	<i>Inches.</i>	<i>Pounds.</i>		
Buffalo (water).....	Rocker dump.....	27	24	25	Electric...	Trolley and storage battery.
Carter.....	Gable dump.....	21	18	12	Horse.....	
Catskill Aqueduct:						
Rondout Siphon.....	Side dump.....	40	30	25	Mules.....	Automatic cage dump. Storage battery.
Wallkill Siphon.....	do.....	40	30	25	Electric.....	
Moodna Siphon.....	End dump.....	40	36	30	Mules.....	
Central.....	Turntable, end dump.	30	24	20	Electric...	Storage battery.
Chipeta.....		20	18		Mules.....	
Cornelius Gap.....		81	36		Electric.....	Derrick dump.
Fort William (water).....		18	18	12	Animals.....	
Gold Links.....	Turntable, end dump.		24	20	Horse.....	Cradle dump.
Grand Central sewer.....	Bucket and flat car.	14	24		Hand.....	
Gunnison, west.....	Side dump.....	35	24	16		Derrick dump.
Gunnison, east.....	Box.....	54	24	16	Electric.....	
Laramie-Poudre.....	Turntable, end dump.	16	18	16	Mules.....	Cradle dump.
Lausanne.....	End dump.....	30	42	30	Electric..	
Los Angeles Aqueduct:						
Little Lake.....	Rocker dump.....	32	24	16	Mules and electric.	Derrick dump.
Grape Vine.....	do.....	32	24	16	Electric.....	
Elizabeth Lake.....	do.....	32	24	35	do.....	Derrick dump.
Lucania.....	Turntable, end dump.	22½	18	20	Horse.....	
Marshall-Russell.....	do.....	28	24	16	do.....	Derrick dump.
Mission.....	Side dump.....	22½	18	25	Electric.....	
Newhouse.....	Box.....				do.....	Derrick dump.
Nisqually.....	Rocker dump.....	27	24	16	do.....	
Rawley.....	Turntable, end dump.	17	18	16	Mules.....	Derrick dump.
Raymond.....	do.....	32	18	16	Horse.....	
Roosevelt.....	do.....	16	18	20	Animals.....	Derrick dump.
Swatch.....	Side dump.....	33	22	30	Electric.....	
Snake Creek.....	End dump.....	20	18	20	Horse.....	Derrick dump.
Stillwell.....	Side dump.....	22	24	30	do.....	
Strawberry.....	Box.....	47	24	25	Electric.....	Derrick dump.
Utah Metals.....	Side dump.....	32	24	34	do.....	
Yak.....	do.....	33	19	30	do.....	Sloping floor.

LOADING MACHINES.

Many attempts have been made to utilize machinery for loading tunnel cars. In several of the larger tunnels, intended for railway purposes, power shovels similar to those used in grading or in open-cut mining have been very successfully used in removing the broken rock of the bench after blasting. The ordinary steam shovel is generally employed, a few minor alterations being made so that it can be operated by compressed air. Power shovels operated by compressed air are also employed in some of the mines in the Joplin (Mo.) district.

The "mucking machine" illustrated in Plate VI was used successfully during the excavation of the Hummingbird Tunnel at Burke, Idaho. Its principal feature is an oscillating trough or shovel armed with teeth and driven by a compressed-air piston in such a manner that the forward stroke is appreciably faster than the return. When in operation the teeth rest upon a steel plate under the muck pile,



"MUCKING MACHINE" USED AT HUMMINGBIRD TUNNEL, BURKE, IDAHO.

and as the shovel is fed forward the broken rock is forced by the jerky motion backward along the trough and discharged upon a belt conveyor which delivers it to an ordinary mine car at the rear. The entire machine is mounted upon a wheeled track or framework and is fed forward by a second compressed-air piston connected with a crossbar that can be jacked against the sides of the tunnel. It is essential that the area of this piston be smaller than the one that drives the shovel, for then if the latter encounters a boulder or other obstruction too solid for it to dislodge, the entire machine can move forward and back with the stroke of the larger piston. By this means the machine is not only prevented from injury before the obstruction is removed, but in many cases it will work the boulder aside without any assistance. One man is required to operate the machine and two more are required to tram the car to and from the end of the conveyor and to shovel the rock out of the corners of the tunnel into the trough, for the machine does not swing from side to side, but merely cuts a swath down the center of the tunnel and hence leaves some material piled on each side. The machine is reported to have reduced the time required to clean the tunnel from six to two and one-half hours and to have made possible a material increase in speed of driving.

A power loader of a somewhat different type was introduced in the excavation of the bench at the Yonkers Siphon. It consisted of a chain and bucket conveyor, similar to that used in mill elevators and on some gold dredges, which delivered the material to a hopper whence it was carried to the tunnel car by a flat endless belt.

Owing to the hardness of the rock and the prevalence of huge boulders, some weighing over a ton and necessitating frequent stops for repairs, this machine was unable to compete satisfactorily with hand loading. The machine was used on the surface for loading rock for use in concrete construction, and at this work it is said to have given excellent satisfaction. The material was taken from the dump pile produced in excavating the heading of the tunnel, in which the rock was broken more uniformly into small fragments than the material produced in blasting the bench. However, the size of this machine precludes its use, without considerable modification, in a small tunnel or heading.

MEANS OF HAULAGE.

In practically all tunnels of any length in the United States, either animals or electric motors have been or are employed to haul the tunnel cars. In Europe, notably at the Simplon and the Loetschburg Tunnels, compressed-air locomotives were used successfully. But although those machines are employed to some extent in this

country in mining and industrial work, they have failed to give satisfaction at tunnels where they have been tried, chiefly because of the cost of high-pressure air, the maintenance of charging stations, the time lost in charging, etc. Many mines also are equipped with cable haulage; but because of the constantly increasing length of haul as the heading advances, the use of this system in tunnel work requires such frequent delays and loss of time in extending the cable system that it is hardly suited for tunnel practice. On the other hand, gasoline locomotives, which have recently proved most successful for coal mining, are in most particulars especially well adapted for tunnel work and deserve full consideration.

The principal advantage of animal haulage is the smaller cost of installation; moreover, it requires no special intelligence on the part of the driver, and the ability of the animals to step across the track at the tunnel headings obviates the necessity of a switch. On the other hand, the costs of maintenance and operation for animal haulage not only are high but go steadily on whether the animal is working or not and are influenced only slightly, if at all, by the size of tonnage handled. For these reasons animals are not economical for use in long tunnels, because the saving in installation expense is soon destroyed by the increased operating costs. Then, too, the odors arising from the track are offensive and disagreeable when animals are employed and their respiration vitiates the underground atmosphere, necessitating more ample ventilation. As far as efficiency is concerned, there is little if any difference between horses and mules, although the latter are considered by some to be the sturdier. Mules, however, are better fitted for work in low tunnels, because they are usually somewhat smaller than horses and, being less nervous, do not throw their heads violently up and back when anything touches their ears.

Electric mine locomotives may be divided into two classes—those operated from a trolley system and those obtaining their electrical current from a storage battery. The former are so familiar as hardly to require description. They generally consist of two motors, ruggedly constructed to withstand rough usage and protected from dust and moisture, mounted upon a cast-iron or structural-steel frame, which also carries the trolley, controller rheostat, and other accessories. The sides of the frame may be placed either inside or outside of the wheels. When placed outside, more space is available for the motors and other equipment and the various parts of the machine are more readily accessible. When the frame is placed inside the wheels the car has a smaller over-all width and is therefore more suitable for narrow tunnels.

The storage-battery locomotive is similar in most respects to the trolley machine, except that provision must be made for carrying the

necessary batteries. In most cases the batteries are carried directly upon the motor itself, but the locomotive installed at the Central Tunnel is somewhat unique, in that the batteries are placed upon a separate car or tender. When the machine is handling cars in the tunnel it obtains its current from the battery; upon reaching the tunnel mouth the tender is left on a side track, where it is accessible for recharging, and a trolley, with which the locomotive is also equipped, is employed for switching.

Electric locomotives are compact and simple in construction and do not emit smoke, gas, or disagreeable odors. They are more rapid and are capable of hauling a much greater load than either a horse or a mule, and the cost of the power used is not nearly so great as the cost of forage. But, on the other hand, they require the installation of extra machinery in the power plant, an expensive trolley wire, or a troublesome storage battery, and the roadbed and track must not only be heavier in construction, but usually the rails must be bonded to make them good electrical conductors. The disadvantage of the cost of the extra electrical machinery is, of course, partly offset by the fact that it can be utilized also to operate the ventilating machinery and to furnish illumination for the tunnel. The use of trolley wires in the restricted tunnel space, however, introduces the grave danger of serious injury to persons coming in contact with them.

Gasoline locomotives consist essentially of a frame, usually of cast iron, upon which are mounted the gasoline engine (usually 4-cylinder), the necessary transmission system containing gears and clutches, together with the carburetor, magneto, cooling system, and other accessories. In external appearance they are not unlike the electric locomotives described above. Two forward and two reverse speeds are usually provided in the machines manufactured in this country, a lower one of 3, 4, or 5 miles per hour and a higher speed double that of the lower. The drawbar pull ranges from 1,000 to 4,000 pounds, according to the size of the locomotives. In some of the machines the exhaust gases from the engine are passed through a tank containing a solution of calcium chloride, which cools the gases and is said to remove all offensive odors from them. In a German-made machine the exhaust gases are sprayed with water to produce the same effect.

The gasoline locomotive combines most of the advantages of both electric and animal haulage. It is self-contained and independent of a central station or any other outside source of power. It is fully as rapid as the electric motor ordinarily used in tunnels and is capable of handling an equal load. The fuel for a gasoline locomotive can be obtained readily in almost any locality, and the machine does not consume fuel when it is not running, a matter of great importance in

tunnel work, where interruptions necessarily occupy a large percentage of the time. Another advantage, although perhaps not so important for tunnel work, is the fact that the haulage system may be expanded by the addition of extra units without alteration in the power plant.

The following table, based upon replies from operators and users of gasoline locomotives received in answer to inquiries sent out by this bureau early in 1912, shows the cost of haulage with these machines:

TABLE 12.—Operating cost of gasoline haulage.

Name of operator.	Months in use.	Average number of trips made by each locomotive.	Average gross tonnage, loaded trip.	Average gross tonnage, empty trip.	Average length of haul, one way.	Average number ton-miles daily.	Fuel used per day.	Cost of fuel per day.	Cost of labor per day.	Cost of lubricating oil per day.	Operating cost, exclusive of repairs.	Cost per ton-mile, not including repairs.	Average daily cost for repairs.	Cost per ton-mile, including repairs.
							<i>Gals.</i>							
Alden Coal Co., Alden, Pa.	5	20	50	13.5	1,750	1,270	9	\$0.86	\$4.48	\$0.20	\$5.54	\$0.004	(a)	
Breese-Trenton Mining Co., Trenton, Ill.	9	30	30	11	2,200	515	20	2.00	5.90	.50	8.40	.016	(b)	
Durham Coal & Iron Co., Chattanooga, Tenn.	9	12	20	6	1,000	59	15	1.65	3.50	.85	6.00	.101	\$1.70	\$0.13
Henderson Coal Co., Pittsburgh, Pa.	7	12	40	12	5,200	615	8	1.04	5.30	.32	6.66	.011	.33	.012
H. B. Swope & Co., Madera, Pa.	11	20	32	10	3,000	475	15	2.25	6.24	.25	8.74	.012	.57	.020
Pocahontas Smokeless Coal Co., Welch, W. Va.	14	15	35	10	2,400	307	12	1.44	4.50	.35	0.20	.02	.25	.021
Midvalley Coal Co., Wilburton, Pa.	12	7	60	27	11,500	1,320	15	1.50	4.14	.12	5.76	.004	.33	.005
Munroe Coal Mining Co., Barnum, W. Va.	5	20	21	6	3,000	300	20	2.20	3.75	.15	6.10	.02	1.00	.024
Roane Iron Co., Rockwood, Tenn.	18	9	36	14	5,800	600	13	1.30	3.50	.41	5.21	.009	(c)	
Shade Coal Mining Co., Mount Pleasant, Pa.	4	12	61	9	2,000	740	15	2.20	2.50	.86	5.56	.0075	.22	.008
Tennessee Consolidated Coal Co., Tracy City, Tenn.	14	35	19.5	7	2,500	440	15	2.50	4.10	.65	7.25	.016	.50	.018
Vaughen Coal & Coke Co., Roderfield, W. Va.	11	10	27.5	14	10,500	830	16	3.60	4.50	.90	9.00	.011	(b)	

(a) None as yet.

(b) Not given.

(c) Not excessive.

Practically the only disadvantage of the gasoline locomotive is the character of the exhaust from the engine, but this can be eliminated by proper and adequate ventilation. If the gas were confined in a small unventilated space the air would soon become unfit for breathing, but as the greater part of the time the motor is traveling back and forth in the tunnel and as a large volume of air is, under proper management, being supplied by the ventilating blower, the exhaust gases from the engine are quickly diluted to harmlessness. It is essential, however, that the blower be arranged to deliver air to the heading through the ventilating pipe rather than through the tunnel, in order that the air may reach the workmen as pure as

possible. Even if it should be necessary to operate the blower at full load, the added cost of doing so would be more than repaid by the saving effected by the gasoline haulage.

DUMPING DEVICES.

The box cars used at the Strawberry Tunnel were dumped by an electrically operated stiff-leg derrick. The hook in the derrick block carried a bail that engaged trunnions, one at either end of the car. The trunnions were placed in such a way that when an empty car was picked up by the bail the weight of the running gear was sufficient to hold the car upright, but if the car was loaded its center of gravity was above the trunnions. A spring-actuated pin, placed in one leg of the bail and engaging a hole in the car body above the trunnion, prevented the car from overturning until it was swung out over the place where the rock was to be deposited, when by pulling a rope the attendant could disengage the pin and permit the car to turn over and deposit its contents. It would then automatically right itself and could be swung back on the track. The derrick was mounted on wheels so that it could more easily be moved ahead, but moving was necessary only at intervals of three to six months.

Among the advantages claimed for this system of dumping was that it could be operated by the train crew, the motorman running the hoist and the brakeman adjusting the bail, thus saving the labor of a dumping gang. Then, too, it gave a much larger dumping area, with a consequent saving of the time that with the ordinary mine car is lost in shifting tracks, etc. But this saving was offset in part by the settling of the dump and on this account the moving of the derrick was accomplished with great difficulty. It is probable that some of this annoyance could be avoided in a future installation by using very wide wheels similar to the type used on roller trucks for moving houses. However, the derrick is expensive, costing when erected at the tunnel approximately \$3,600, of which hardly more than \$1,500 could be realized from its sale after the completion of the work. For this reason its use must extend over a considerable length of time in order that the saving in wages may repay the original cost.

At the Newhouse Tunnel the loaded cars were rolled into a cylindrical steel framework having rails at the bottom and a set of angle-iron guides at the top with just enough clearance space between them to hold the car firmly. The entire apparatus was then revolved by an electric motor until overturned, emptying the contents of the car, and it was then righted by continued revolution and the car removed. Although used here only for ore cars, the dumped mate-

rial falling into a bin for shipment, it offers a satisfactory and reasonably inexpensive means of dumping the more durable solid-body and truck cars and could doubtless be applied to tunnel dumps by the use of a light trestle or similar structure.

Almost any one of the various cradle dumps used at many coal mines can readily be adapted to tunnel work by mounting it upon a stout frame of logs or large timbers, which could be pushed forward along the top of the rock pile, as necessary. By this means it is possible to eliminate hinges and turntables between the body and the truck of the car, thus simplifying and strengthening its construction. One of these cradle dumps was used at the Lausanne Tunnel. It was not expensive and saved considerable time in dumping cars and in keeping the rock pile in proper condition. It was pushed forward by the motor every two or three days, requiring only a few minutes for the operation. Figure 22 illustrates a similar dumping device used at the Cameron mine, Walsenberg, Colo. It has the added advantage of being mounted on a turntable, thus

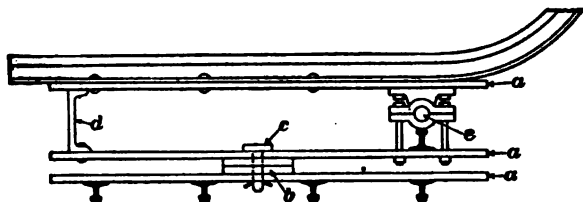


FIGURE 22.—Turntable cradle dump used at Cameron mine.

giving nearly double the top width of dump attainable with ordinary cradle devices. As described in *Mines and Minerals*,* the dump consists essentially of three plates of $\frac{1}{2}$ -inch iron 3 by 4 feet in size. To the top plate are bolted a pair of mine rails with the ends bent up into horns. This upper plate revolves on the mine car axle *c*, the bearings for which are supported upon a mine rail and bolted to the middle plate. A piece of channel iron, *d*, is bolted to the middle plate, and upon it the dump falls back after a load of rock has been discharged. The upper plates as a unit revolve upon the two annular pieces of iron, *b*, 22 inches in diameter. The king-pin, *c*, is 1 inch in diameter and the plates, where it passes through them, are reinforced by a piece of $\frac{1}{2}$ by 3 inch bar iron. The lower plate is supported by four short lengths of 12-pound mine rail.

INCIDENTAL UNDERGROUND EQUIPMENT.

TUNNELING MACHINE.

Although tunnels have been constructed for mine drainage, irrigation, and supplying water to cities for thousands of years, they were so few in number during ancient times and constructed at such

irregular intervals that there was no great incentive to improve upon the methods ordinarily employed in building them. However, with the advent of the steam railroad it was soon realized that the desirability of maintaining level gradients would necessitate the driving of many tunnels, and the active minds of inventors were immediately directed toward the problem of making a machine that would do this work more or less automatically.

The first tunneling machine of which any record could be found was constructed at Boston in 1851 for use in the Hoosac Tunnel. It weighed 70 tons and was designed to cut in the face of the tunnel a circular groove 13 inches wide and 24 inches in diameter by means of revolving cutters. The trial of this machine in the tunnel proved unsuccessful, and a distance of only about 10 feet was cut with it before it was abandoned. In 1853 the Talbot tunneling machine, which was designed to make an annular cut 17 feet in diameter, leaving a cylindrical core to be removed by blasting, was tested near Harlem, New York, but also proved unsuccessful. Later a smaller machine was constructed, adapted to cut an 8-foot annular groove. The smaller machine, although less unwieldy than its predecessor, also proved a complete failure after \$25,000 had been expended upon it. Numerous machines constructed upon almost every conceivable principle have been experimented with since 1853, but most of them have been entirely discarded.

However, it is not safe to predict that a tunneling machine will not be constructed to perform this work in the future because, difficult as the problem of designing such a machine appears, the obstacles in the path are no greater than they have been in scores of other instances where slow and costly manual methods have been superseded by less expensive and more expeditious mechanical processes. The invention of some new rock-cutting device, or the material improvement of some of those now known, may simplify the problem to such an extent that the construction of a successful tunneling machine will be rendered comparatively easy. The simple device of putting an eye in the point of a needle made the sewing machine possible; the breech-loading gun was a complete failure until the brass cartridge was invented; and not even the genius of a Langley or a Wright could construct a flying machine until the internal-combustion engine had reached its proper stage of development.

One of the great obstacles encountered by legitimate investigators in this field has been the great difficulty of obtaining funds, for with this machine, as well as with any other new and complicated machine built to operate under difficulties and strains that can not be measured in advance, costly experimental work is necessary. It must be remembered that machines of the size and strength necessary to cut the entire face of a tunnel as a single operation are of necessity costly,

and their maintenance during the trial stages is extremely expensive. For this reason success can hardly be expected unless the inventor, or the company back of him, commands large funds. The failure of one badly designed and inadequately financed machine after another and the suspicions aroused in the minds of possible investors by the untruthful and flamboyant "literature" that has been issued by too many alleged tunnel-machine companies in their efforts to "work the public" have caused most people to look upon machines of this kind with extreme distrust, much of which is indeed just, for even a casual scrutiny of the claims of many of these concerns shows clearly the fictitious character of the statements.

ILLUMINATION.

With few exceptions, illumination for tunnels and adits in the United States at the present time is furnished by electricity, acetylene gas, or candles. The smoky open-flame miners' oil lamp is occasionally used in tunnels situated in the coal-mining districts, and, of course, under conditions that prohibit the use of an open flame, safety lamps must be employed. When acetylene gas is employed it is usually generated in portable lamps, but during the work on the water conduit for Washington, D. C., in 1899, this gas was manufactured at a plant on the surface and carried by pipes underground, where it was burned in jets at regular intervals. Coal gas was similarly employed at the Mount Ceniz Tunnel, which was started in 1857 and opened for traffic in 1872. The following table, however, shows the present practice with regard to means of illumination:

TABLE 13.—*Means of illumination at various tunnels.*

Name of tunnel.	Method of illumination.
Buffalo (water).....	Electric lamps.
Carter.....	Acetylene lamps and candles.
Catskill Aqueduct.....	Electric lamps at intervals and usually a cluster of lamps in the headings.
Central.....	Acetylene lamps.
Fort William (water).....	Electric lamps (16 candlepower) every 75 feet and one 32-candlepower in heading.
Gold Links.....	Candles.
Gunnison.....	Electric lamps, cluster in heading, and candles.
Joker (drainage).....	Electric lamps.
Laramie-Poudre.....	Acetylene lamps for drillers, candles for muckers.
Lausanne.....	Miners' oil lamps and safety lamps.
Los Angeles Aqueduct.....	Electric lamps and candles.
Lucania.....	Acetylene lamps.
Marshall-Russell.....	Do.
Mission.....	Electric lamps every 200 feet, cluster in heading, candles.
Newhouse.....	Electric lamps at stations; acetylene lamps in heading.
Niequally.....	Electric lamps every 75 feet; cluster in heading.
Ophelia.....	Candles.
Raymond.....	Electric lamps every 200 feet; cluster in heading.
Rawley.....	Acetylene lamps.
Roosevelt.....	Electric lamps.
Siwatch.....	Electric lamps every 200 feet and candles.
Snake Creek.....	Acetylene lamps.
Stillwell.....	Electric lamps in heading, candles.
Strawberry.....	Electric lamps every 135 feet; cluster in heading.
Utah Metals.....	Electric lamps at switch; acetylene lamps in heading.
Yak.....	Electric lamps.

Neither candles nor the open-flame oil lamp can be recommended as a means of lighting a tunnel or adit during construction. Practically all that can be said in their favor is that they require a much smaller initial outlay than electricity or acetylene, yet they are more expensive per unit of light than either acetylene or electricity, consume a greater amount of oxygen, and give off a correspondingly greater amount of noxious gases. Candles not only do not give enough light, but what they do supply is flickering and unsteady unless there are no drafts, and as they are quickly extinguished by the exhaust blasts from air drills they can not be placed to light properly the work of the drillers; hence with their use the efficiency of a high-priced drillman is greatly reduced. Candles are often wasted or dropped into the muck pile, an item of loss that may amount to a considerable sum in the long run. The open-flame oil lamp can not be prevented from giving off soot and smoke, which obscure the light thrown on the work, and the soot, collecting in the miner's throat and lungs, irritates the mucous membranes and renders them easily susceptible to disease.

Electric incandescent lamps possess a number of advantages for tunnel work. They give a brilliant and steady light, one that is not affected by drafts and neither pollutes the air with soot nor vitiates it by consuming the oxygen. By combining several of them in a cluster plenty of light in the heading is obtained for the drillers and shovelers, tending toward efficiency. To offset this advantage, however, the fact remains that unless they are used in connection with electric locomotives, drills, or similar machinery, the cost of lamp installation is almost prohibitive; even with the electric appliances in use, the extra wiring and the lamps themselves are expensive, the lamps entailing considerable loss through breakage. Electric lights are also at a disadvantage because they are not easily portable, and the removal and replacement of bulbs and wires in the heading before and after blasting complicate an already involved situation. Moreover, this means of illumination is uncertain, especially in wet tunnels, because the chance occurrence of a short circuit through moisture, accident, or carelessness throws the entire work in darkness, and, if other means of lighting are not at hand, stops all work until the trouble can be remedied. Again, whereas the use of electricity underground is always attended with some danger, this is especially true in the case of lighting appliances. The supposition is that the wires are protected, but the rough usage to which they are subjected soon destroys insulation, rendering persons who handle them (as they must do frequently) subject to severe shock.

One is tempted to say that the ideal means of tunnel illumination is found in the portable acetylene lamp, combining as it does the

advantages of other illuminants while avoiding most of their defects. It may be obtained on the market to-day in a number of different designs and sizes adapted for practically every kind of work; the one most generally observed at the tunnels visited was about the size of an ordinary fruit can and capable of burning for 8 to 10 hours on one charge of carbide and water. Although too large for use on a cap, it was provided with a hook so that it could be suspended from any convenient place. Lamps suitable for wearing on a miner's cap are obtainable and will burn for two or three hours without recharging—an operation easily performed in two or three minutes. The initial expense of an acetylene lamp is not high and, with the possible exception of the electric arc, it furnishes the brightest known artificial light used for underground work, consuming only one-fifth as much oxygen as candles. It is ordinarily provided with a reflector, which not only concentrates the light upon the work where it is needed, but shields the flame from drafts so that it will burn steadily unless placed directly in front of the exhaust from an air drill. Extensive use by some of the larger mining companies in this country has shown that the cost of the carbide is much cheaper than either oil or candles, the use of acetylene lamps cutting the cost of light practically in half. At the Saginaw mine, Menominee Range, Mich., the cost is reported as only 2 cents per shift of 10 hours. Such lamps require practically no attention, are completely portable, and are not subject to breakage as are incandescent lamps. By giving the workman plenty of light his efficiency is not only increased but he is better able to guard against the dangers of underground work, such as an insecure roof, an unexploded stick of dynamite in the muck pile, or any other of the many dangers to which he is at all times exposed.

TELEPHONES.

Although it has been repeatedly stated in newspapers, engineering periodicals, and even by State legislatures that every mine should be provided with a telephone system, the importance of telephones in tunnel work can not be too often reiterated, not alone because of the greater safety they insure but on the ground of efficiency and economy as well. The sources of accident in tunnel work are too numerous to mention—falls of roof, caves, premature or delayed explosions, flooding, and noxious gases being some of the more common. When an accident occurs in a tunnel that is equipped with a telephone system, not only can assistance be summoned quickly, but provision can be made beforehand for the care of injured men upon reaching the surface; if professional help can be summoned and due preparation made while the men are still on the way from the heading, invaluable time is saved, for there are many instances where prompt medical attention has decided the question of life or death. Then, too, failure to obtain

a proper round of holes in the given time, difficulty in blasting them to the full depth, or any of the many problems that commonly arise in tunnel driving call for a decision on the part of the foreman as to the method of procedure. Ordinarily the man intrusted with this position is capable of meeting such conditions as they arise, but it stands to reason that the work of the shift will be more efficient if the foreman can be in touch constantly with the superintendent and when in doubt receive suggestions and advice from the more experienced man's better judgment. Delay can be avoided in good part if the tunnel is equipped with a telephone, because the necessity that involves sending for fresh materials, tools, powder, etc., can either be foreseen and provided for promptly from the outside without the loss of a man from the heading crew, or when unexpected emergencies arise only half the usual time is necessary to obtain the needed supplies. Causes of accident and delay can not always be foreseen, it is true, but they can be met promptly and further damage to men and property can be prevented by the use of the telephone; that these advantages are appreciated is shown by the fact that most of the tunnels and adits examined in the field were equipped with telephones.

The type of telephone equipment should be carefully chosen because every telephone is not suited for underground use. For use in tunnels the instrument must be waterproof, dustproof, and to be useful, it must be placed as near the heading as possible; it must be designed to withstand the frequently recurring concussions of blasting. In the most successful types of mine telephones the mechanism is placed in a heavy metal casing in such a way that the essential parts are instantly accessible upon opening the outer door, but are tightly sealed when it is closed. The more delicate mechanism is guarded further by an inner door, also of iron, and the wires are protected so that water can not enter the casing. The bells must necessarily be placed outside, but they are protected by a metal hood, which, however, does not prevent their being heard for a considerable distance.

The telephone line for tunnel work is somewhat simpler than a similar line on the surface, because no poles are required and the wires can be strung from ordinary glass or china insulators fastened to plugs in the roof or to light cross timbers. Ordinary bare iron wire can be used, but much better results are obtainable where rubber-covered wire is employed; and for the same reason a full metallic circuit is desirable, although the telephone may be operated with only one wire by using a ground connection for the return. But as the usefulness of a telephone system is measured entirely by its reliability, the best is in the end by far the cheapest.

To guard against noise and the effects of concussion the telephone should not be placed nearer the heading than several hundred feet. Although such an installation is convenient for anyone in the tunnel desiring to call up the office, it makes difficult and sometimes even impossible the obtaining of any response to a call originating on the surface. To obviate this difficulty, the use of an extension loud-ringing call bell is recommended, which, if placed behind a jutting rock or in some similar protected position, apprises the foreman at the heading instantly of any call at the telephone. Such a bell should be connected with the telephone circuit by a flexible insulated cable mounted upon a reel in such a way that the bell may be advanced regularly to keep pace with the tunnel progress and need be never farther than 200 feet from the heading. When the cable is extended to full length, perhaps 1,000 feet, the telephone should be advanced to a point as near the heading as practicable and the extra cable reeled up once more.

INCIDENTALS.

Among the many devices used to save time and promote efficiency underground are the hose supporter and the drill rack, both of which can be made readily by any tunnel blacksmith. The hose supporter consists merely of two telescoping pieces of iron pipe, the length of each being about three-fourths of the width of the tunnel. In operation the hose is placed over the pipes, which are then extended until their pointed ends fit into convenient niches on either side of the tunnel near the roof; the pipes are clamped into position firmly by a threaded key which is provided for this purpose. By using two or three of these spreaders the hoses are kept clear of the shoveler, who is thus saved no little trouble and annoyance and is able to work to better advantage. The drill rack is simply a rack for separating different lengths of drill steel. A satisfactory form consists of an A frame made of 4 by 4 inch timbers, into which iron pegs are driven at convenient intervals. The segregation of the sharp drill steels on this rack enables the helper to pick out the proper length with assurance and dispatch.

TUNNEL-CONSTRUCTION METHODS.

The following discussion of methods of tunnel construction is restricted chiefly to those in which the entire cross section is excavated in one operation. The majority of tunnels and adits driven for mining work and many tunnels intended for irrigation and water supply are small enough to be driven in this manner; but in the construction of the larger undertakings, such as are required for railroad or similar purposes, it is customary to drive a pilot tunnel or heading, as it is sometimes called—although the term is also employed to designate the advancing end of any tunnel—previous to the main body of the work which then consists in enlarging the smaller excavation to full size. The latter method, in addition to lowering the average cost of the entire work, because the process of enlarging is much easier and less expensive than that of driving the heading, also gives a valuable preliminary insight into the conditions that must be encountered later by the main tunnel, and enables the constructor to anticipate emergencies and make provision for them in his plans, thus aiding to prevent accidents and loss. However, as this bulletin is to be confined chiefly to mine adits and small tunnels, a discussion of the various phases of the “heading-and-bench” system can not be treated here as such, although the methods used in excavating in one operation the entire section of a small tunnel are in most cases applicable to the driving of headings for larger tunnels. Local conditions at each project necessarily modify methods to such an extent that it is impossible to make a general analysis to fit all cases, but the discussion following is intended to bring out some of the more important features of the methods employed in the various operations of drilling, blasting, mucking, and timbering as they are applied to the driving of mine adits and tunnels.

The methods employed in the operation of drilling, aside from the mere running of the machine, are concerned chiefly with the number of drilling attacks, the mounting of the drills, and the number, depth, and direction of the holes. The choice of ammunition, including the fuse and caps as well as the explosive, and the manner of loading and firing the holes are the main factors influencing the efficacy of blasing. Considerations of safety, however, demand that suitable provision be made for storing the explosive and thawing it as needed and that proper precautions be observed in its use. The number of men and the positions in which they work, the method of handling the muck cars in the heading, and the use of steel plates

upon which to shovel are the salient features of mucking, whereas timbering is concerned chiefly with the materials employed and the different types or methods of using them.

DRILLING.

NUMBER OF SHIFTS.

Relative to the number of shifts preferable for drilling, one of the chief advantages claimed for the single shift per day is economy. By having the débris cleared from the heading by the shoveling crew at a separate time, the drillmen, upon reaching the face, are able to start immediately to work setting up the machines and preparing to drill the round; there is therefore no waste of time or labor on the part of these men or the helpers in shoveling out débris preparatory to mounting the drills. This method is especially economical when vertical columns are employed. During the process of drilling, the operators and their helpers are not interfered with or hindered in any way by the shoveling crew, and there is therefore a saving of that loss of motion that can hardly be prevented when two crews are working simultaneously in the heading. Moreover, as there is no delay in getting started, the round of holes can ordinarily be completed within the allotted time, and even if this can not be done plenty of extra time is available without delaying the following shift. The drilling and mucking shifts can be distributed so that there is no loss of time and wages while the men are waiting for smoke and gases produced in blasting to be removed from the tunnel—of cardinal importance where the provisions for ventilation are inadequate. These considerations go to support the contention that the actual excavation cost per foot of tunnel is lower with this method than with other systems.

On the other hand, by employing a single drill shift the daily progress in driving the tunnel is necessarily limited to the advance gained from the one attack, and therefore the completion of the work must inevitably be delayed. Most tunnels are practically worthless until completed. If their construction is not pushed as rapidly as possible not only is the capital invested in the equipment, tools, etc., tied up much longer than necessary and the cost for interest and the depreciation charges proportionally increased, but there is also a delay in the realization of the benefits to be derived from the tunnel, which in most cases is more than sufficient to offset any saving in excavation cost. For example, if an adit is being driven to drain a mine the extraction of additional ore below water level is greatly delayed; or if the adit is intended to lower the cost of transporting the ore to the surface the loss on the additional tonnage handled in the old way, owing to the delay in its completion, should be

charged against this system of operation. Similarly, with an irrigation tunnel, the entire season's crops may be lost from the longer time required to complete the tunnel if it is excavated by the one-shift method. Then, again, the cost for administration and many other of the fixed charges are operative during the period of construction, independent of the number of shifts per day, and as the daily progress increases with the additional attacks per day, the proportionate charge against each foot of tunnel driven will be smallest when the greatest number of shifts are employed. Although by a saving of the time and wages of workmen there is a seeming economy in the cost of excavation by the one-shift method, when factors that reach deeper are considered, in most cases the ultimate cost of the tunnel will be lowered by methods that make for speedier completion.

Greater progress is undoubtedly attained with two shifts per day than with one, and if the work is properly organized there need be only little added excavation cost. The usual custom with this system is to have the shovelers start work somewhat in advance of the drillers and to work at first in removing the broken rock directly at the face, so that the drillers may set up their machines promptly. At some adits and tunnels where two drill shifts were used the drilling and mucking took place simultaneously, the drillers themselves attending to the work of clearing out for the set-up. Of these two methods the former is preferable, not only because it economizes the time and exertion of higher-priced men but also because the length of time when both crews are at work together in the heading—and consequently the inevitable amount of interference and interruption—is thereby lessened. At a few places three crews of shovelers were required to remove the rock broken by two drilling attacks. This system is obviously expensive because the cost of the extra shovelers must be charged against a footage only slightly, if at all, increased by their efforts, and it entails for two of the three shifts the disadvantage of simultaneous work just mentioned; it is therefore not desirable.

The consensus of usage at tunnels and adits where the best results in driving have been achieved, both in this country and abroad, leads to the conclusion that the three-shift system of attack is the most desirable. This method has a number of opponents who claim against it four chief disadvantages: (1) That time is lost on the part of the drillmen in getting the machines set up and in operation; (2) that the greater number of men crowded in the restricted space of the heading are in each other's way and therefore unable to work to the best advantage; (3) that the men must be paid for time wasted in waiting for the smoke and gases produced in blasting to be cleared

from the heading; (4) that the system makes no provision for delays due to adverse conditions. As is pointed out subsequently, the time consumed in setting up the machines can be made negligible by the use of suitable methods of mounting and by properly directing and blasting the round of holes. Some crowding is, of course, unavoidable, but it is more than offset by the gain in efficiency resulting from the various incentives that can result only from the three-shift method. To begin with, the shovelers have constantly before them the necessity of removing the waste rock before the drillers have finished their work and are therefore unconsciously speeded up by the competition. At the same time the drill men endeavor to have their holes finished by the time the tunnel is cleared in order that no delay may be attributed directly to them. Both crews are inspired to better work by the knowledge that a competing shift is to follow immediately upon their heels, taking their places and performing similar work. Then, too, after the holes have been drilled the extra men from the shoveling crew are of great assistance in taking down the machines and removing them and their mountings, hose tools, and other articles that must be taken to a place of safety during blasting. As to the time wasted in clearing the tunnel of smoke, if the tunnel is adequately equipped with ventilating apparatus this operation should require little more than 15 minutes—just long enough for the men to eat their lunches—time that would have to be lost at any rate. Delays, of course, can not always be prevented, but the men are encouraged by rivalry to reduce these to the minimum, knowing that their work is to be compared with that of the shift to follow. These answers to the various objections are in no sense theories, but are deductions from actual observation and a study of conditions as they existed at tunnels where some of the most efficient work in this country was being performed.

The ideal results of the three-shift method, to be sure, are obtained only through perfected organization and good management, but they utterly disprove the contention that efficient work is not possible under those conditions. That the method may produce the most rapid progress has never been gainsaid, and with proper handling the actual cost of excavation per tunnel-foot need be little, if any, greater than with other methods; moreover, as has been shown, in most cases the system affording greater speed is within limits ultimately the more economical one. For these reasons, unless the conditions are indeed exceptional, the employment of three drilling shifts per day is recommended, and the following discussion of other phases of tunneling methods will, unless otherwise noted, be predicated upon the assumption that three drilling shifts are being employed.

METHODS OF MOUNTING DRILLS.

American tunnel practice is almost equally divided between the horizontal-bar and the vertical-column methods of drill mounting. The horizontal-bar mounting consists essentially of an iron pipe, 4 to 6 inches in diameter, a little shorter than the average width of the heading, and provided with a solid head at one end and a jack-screw with a capstan head at the other. The vertical-column mounting, which is rarely employed with more than two drilling attacks per day, is not greatly dissimilar except that it is usually provided with a yoke and two jackscrews at one end, and its length is somewhat less than the height rather than the width of the heading. In several notable European tunnels a drill-carriage was employed, however, so that a discussion of this method of drill mounting should not be omitted.

The horizontal-crossbar method of mounting rock drills can perhaps be best illustrated by a description of the procedure used at the Laramie-Poudre Tunnel. As soon after the blasting as ventilation permitted (ordinarily 10 to 15 minutes), the workmen returned to the face from a position of safety 1,500 to 2,000 feet away, bringing with them an ordinary tunnel car containing the crossbar, drilling machines, tools, hose, etc. The three drillers, sometimes with the assistance of the foreman, first removed from the roof or walls loose rocks that might have fallen later, possibly causing injury. This accomplished, they next cleared a space in the top of the rock pile, for 2 or 3 feet back from the face of the tunnel and perhaps 4 or 5 feet from the roof, in order that they might have room to work when drilling. Because of methods of blasting especially employed for this purpose the rock pile usually occupied only a small part of this space, so that ordinarily little work was required to clear it out. In the meantime the helpers were expected to unload the bar and machines from the car, placing them on the rock pile conveniently at hand, and connect the hose to the air and water mains. As soon as a proper space was cleared out the bar was picked up by the drillers and helpers and held in position transversely across the tunnel at a measured distance from the face and roof, as directed by the foreman, where it was blocked, wedged, and finally screwed as tightly as possible in place. The drillmen then placed the machines upon the bar and started drilling as soon as the helpers completed connecting the hose to the drills. The necessary holes having been drilled from this position of the bar and the waste rock having been removed in the meantime by the shovelers (an operation carried on simultaneously with drilling and ordinarily accomplished before the drillers had finished), the machines were taken off, the bar lowered and set up again about 18 to 24

inches from the floor, the drills replaced, and one or two holes drilled by each machine from this position of the bar. The machines and the bar were then placed in a tunnel car and removed from the heading during the blasting. This method, sometimes slightly modified, was used at several other tunnels and adits with almost equally good results.

The procedure with the vertical-column method of mounting is similar to this in some respects, but there are also some important distinctions aside from that of upright position. Owing to the vibration produced by the drills, neither method of mounting will give satisfaction unless the bar is firmly jacked against solid rock. The vibration is intensified and the need for a substantial foundation is much greater with the vertical column, because the drills are usually mounted on cross arms projecting from the columns at right angles, thus affording a leverage for any movement of the drill. It is therefore necessary to remove all of the waste rock from the space immediately in front of the face of the tunnel prior to drilling, in order that the foot of the column may rest upon the solid floor, which, at the two or three tunnels where this method was employed with the three-shift system, caused considerable delay even under normal conditions. But in the majority of places where this method of mounting was employed not more than two drilling attacks were attempted per day, and the extra work of clearing away was performed by the crew of shovelers before the drillers started work.

The best results with the carriage mounting for drills were obtained during the construction of the Loetschberg Tunnel through the Bernese Alps. In the first type of carriage employed there, the horizontal bar carrying the drills was mounted at the end of a steel beam which was pivoted to a truck and counterbalanced at the other end by a heavy weight. Before this carriage could be brought sufficiently near to the face, even with the long beam, for the crossbar to be jacked in position, it was necessary to clear a rather large passageway through the center of the rock pile down to the floor. In doing this, part of the material was carried away and the remainder piled on either side of the tunnel to be carried away during drilling. When the passage was finished, however, the carriage with the crossbar and drills mounted upon it and extending longitudinally was quickly rolled to the face, the bar swung around and jacked into position, and the drills were at once started to work.

This carriage was superseded by one in which the counterbalanced beam was abolished and the drill bar carried directly upon a short post mounted on the truck. With this device practically all of the broken rock had to be removed from the heading before the carriage was brought to the face, after which, however, the drills started at work promptly.

One of the most important factors to be considered in choosing a method of mounting for tunnel work is the time required to get the drills in operation after blasting, including not only the actual time employed in setting up the necessary apparatus, but also the time consumed in the preparatory work of clearing away *débris*. The time spent in waiting for the smoke to clear is, of course, independent of the method of mounting and can therefore be ignored in this connection. With the horizontal-bar system used at the Laramie-Poudre Tunnel, the time normally employed in mucking back was rarely more than 15 to 20 minutes. Jacking the bar in place occupied 5 to 10 minutes and attaching the drills and making the water and air connections usually required 10 to 15 minutes. The entire operation thus consumed, under ordinary conditions, 30 to 45 minutes, but it was not at all unusual for the drills to be in operation within 20 or 25 minutes from the time the drillmen reached the heading. At other tunnels and adits using this system the time required for similar work was reported as 30 to 60 minutes. Owing to the much greater quantity of material to be cleared out when the vertical-column method is employed, the time consumed in getting the drills in position to start work at adits and tunnels where the three-shift system was used ordinarily ranged from $2\frac{1}{2}$ to 4 hours, and even under the most favorable circumstances was rarely less than 2 hours.

The time spent in the Loetschberg Tunnel in removing the waste rock was approximately $1\frac{1}{2}$ hours with the first type of carriage used and $1\frac{1}{2}$ to 3 hours with the later model, but in order to attain such speed nearly twice as many men were employed at the work as are usually found in American tunnel headings. After the Loetschberg Tunnel had been cleared of the necessary amount of *débris*, however, the machines could ordinarily be started in 5 to 10 minutes.

Aside from the question of the time consumed in clearing, the amount of waste to be removed has another bearing on the problem of choosing a mounting. In order that there may be no delay in getting the drills at work, usually the attempt is not made to remove the waste rock entirely from the heading before setting up the mountings, much of it being merely shoveled to one side and removed later. This preliminary work is often performed by the drillmen, especially with the three-shift system; and where (as in the case of the vertical-column method) there is a great deal of it to be done, by the time they have the machines set up and are ready to start drilling these men are pretty well tired out and consequently can not work as rapidly and efficiently in drilling the required holes. Even if the work is performed by the regular shoveling crew, the men certainly are not stimulated by the knowledge that they are performing dead work, and that every shovelful handled in clearing back must be moved again later. This disadvantage obtains not only in the three-shift

system but in many cases where two shifts are employed and the shoveling crew start ahead of the drillmen and commence work clearing away the face for a vertical-column set-up. The horizontal-bar and, to a lesser degree, the drill-carriage methods have the advantage of requiring a much smaller proportion of duplicated work.

The adaptability of the mounting for the work required of it after the drills are in operation is another factor to be reckoned with. The advocates of the vertical-column method claim that it enables the holes to be placed to better advantage, and this is quite truly the case when piston drills are employed. But hammer drills mounted on a horizontal bar can place the holes just as effectively, if not more so. With either type of machine the drill carriage is badly handicapped. It was discovered with those used in the Loetschberg Tunnel—and the same disadvantage was experienced at an adit in this country where a similar drill carriage was tried and soon abandoned—that it was impossible to point the inclined holes in such a way as to obtain the maximum efficiency from the explosive used. Therefore, in order to make the holes break to the bottom it was necessary to use heavier charges of explosive, and the holes were not drilled as deeply as they might otherwise have been. The shallower holes necessitated greater labor in the unproductive preparatory work of setting up and tearing down the drills and increased the opportunities for delays in blasting. Then, too, it is impossible with one set-up of a horizontal bar, such as was used in the carriage method of mounting, to make the holes near the bottom of the tunnel sufficiently horizontal to insure an even floor, necessitating trimming, and causing trouble in maintaining the proper tunnel grade.

The fact must not be overlooked, however, that with the carriage method the drills are subject to less wear and tear because they are kept on the bar continually and are not thrown around on the floor and muck pile. When this is permitted the drills are apt to become filled with sand, grit, etc., which, because of friction and abrasion, increases the cost of repairs. Nor should the facility in changing to a new hole possessed by the horizontal bar and the drill carriage be disregarded. When these methods of mounting are employed, all that is necessary in starting a new hole is to slide the drill along the bar and clamp it in place, but with the vertical column not only the machine but the cross bar as well requires adjusting; as the adjustment is vertical instead of horizontal, the entire weight of both drill and cross arm must be lifted or sustained at nearly every change.

Taking into consideration all of the factors mentioned, the horizontal bar proves to be the method of mounting drills best adapted for tunnel work. Its use enables the drills to be put in operation with the least loss of time and by the smallest number of men. It

requires the rehandling of the minimum quantity of waste rock so that the drill men are not fatigued before they start drilling or the shovelers disheartened by dead work. It permits directing the holes in such a way that the maximum strength of the explosive is utilized, drilling deeply so that too great a part of the time need not be spent in preparatory work, and placing the holes to insure the breaking of the roof and floor smoothly and at the desired grade. It is especially adapted for use with the more rapidly drilling hammer machines and lends itself readily to removal when necessary. In common with the vertical type it is subject to the danger of allowing grit to become lodged in the machines, but this can be partly prevented by care in handling.

PROPER NUMBER OF HOLES.

Any determination of the proper number of holes to be used in driving a tunnel or adit of a given size is dependent upon several factors. A large number of holes in which a large charge of explosive may be placed expedite the operations of driving, because the heavier blast tends to hurl the rock farther away from the face, not only saving time in setting up the machines but also giving the shovelers more room and enabling them to work to better advantage on more widely scattered material. But at the same time, holes that are not strictly a necessity entail an extra expense not only for the explosive used in them, but also for the time required in drilling, especially if the drilling work requires more time than the operation of removing the rock, as any extra holes delay both crews. If the proper number of holes is being used, the major part of the rock should be broken into fragments small enough to be shoveled readily, although an occasional boulder, because of the relaxation it affords the workman from the steady grind of shoveling is said to expedite rather than retard the speed with which the spoil can be loaded into the tunnel cars.

The central factor, however, in a just determination of this question is undoubtedly the physical character of the rock being penetrated, which is never twice alike in different localities, preliminary experimentation being usually necessary in order to discover what number of holes will produce the best results. Generally speaking, igneous rocks require more holes than sedimentary rocks, but there are wide divergences in both classes. The holes must be more closely spaced for a tough rock that is close grained and massive than for one that is brittle and easily shattered, even though it may be harder and more difficult to drill. Bedding or joint planes or joint cracks are of great assistance, and a rock in which they occur will be more easily broken and hence require fewer holes. The following

table shows the number of drill holes used in American tunnels penetrating different classes of rock:

TABLE 14.—*Number of holes used in driving tunnel headings in various American tunnels.*

Name of tunnel.	Number of holes.	Character of rock penetrated.	Approximate area of heading.	Square feet of heading per hole.	
				Sedimentary rocks.	Igneous rocks.
Burleigh.....	16	Granite and gneiss.....	42		2.6
Buffalo (water).....	22	Limestone.....	120	5.5	
Carter.....	10-11	Gneiss, granite, and porphyry.	41		3.7-4.1
Catakill Aqueduct:					
Rondout Siphon.....	22	Limestone, sandstone, and shale.	120	5.5	
Wallkill Siphon.....	24	Shale.....	120	5.0	
Moodna Siphon.....	24	Sandstone and shale.....	120	5.0	
Yonkers Siphon.....	21	Gneiss.....	120		5.7
Central.....	18-24	do.....	35		1.5-1.9
Chipeta.....	15-19		57		3.0-3.8
Fort William (water).....	14-20	Basalt.....	35		1.7-2.5
Gold Links.....	12	Gneiss and granite.....	48		4.0
Grand Central sewer.....	18	Gneiss.....	40		2.2
Gunnison.....	24	Altered granite.....	60		2.5
Joker.....	19-21		130	6.2-6.9	
Laramie-Poudre.....	21-26	Close-grained granite.....	70		2.7-3.3
Lausanne.....	15-21	Shale, conglomerate, and coal.	85	4.0-5.6	
Los Angeles Aqueduct:					
Elizabeth Lake.....	25	Granite.....	145		5.8
Little Lake division.....	14-16	Medium hard granite.....	90		5.6-6.4
Grape Vine division.....	20-21	Hard granite.....	90		4.3-4.5
Lucania.....	25	do.....	65		2.6
Marshall-Russell.....	18-20	Granite and gneiss.....	72		3.6-4.0
Mission.....	12-14	Shale and slate.....	37	2.6-3.1	
Newhouse.....	19	Gneiss.....	65		3.4
Niaqually.....	18	Rhyolite.....	95		5.2
Northwest (water).....	22	Sedimentary rock.....	110	5.0	
Ophelia.....	20-24	Granite.....	80		3.6-4.0
Rawley.....	25-27	Andesite.....	55		2.0-2.2
Raymond.....	14	Gneiss and granite.....	80		5.7
Roosevelt.....	24-26	Hard granite.....	60		2.3-2.5
Siwatch.....	12	Granite.....	45		3.7
Snake Creek.....	16	Diabase.....	65		4.0
Spiral.....	21	Limestone.....	175	8.4	
Stillwell.....	16	Conglomerate and andesite.....	50	3.1	
Strawberry.....	16-18	Limestone, sandstone, and shale.	50	2.8-3.1	
Utah Metals.....	12-16	Quartzite.....	80	5.0-6.6	
Yak.....	18	Limestone, sandstone, shale, and granite.	50	2.8	

DIRECTION OF HOLES.

Chiefly because of the great influence of local conditions, the arrangement of drill holes is rarely identical in driving at any two tunnels. For reasons to be explained later, however, it is customary to drill a part of the holes (called the "cut" or "cut holes") in such a manner that when blasted they will remove a core of rock from the face and thus decrease the work to be done by the remaining holes. Practically all of the various means of arranging drill holes in the headings of American tunnels may be summarized as follows into three main types according to the kind of cut employed.

The wedge or V cut is the one most commonly employed in tunnels in this country. It consists essentially of several pairs of holes drilled from opposite sides of the heading in such a manner that when properly charged and exploded they will break out a wedge-shaped core of rock, usually extending from the roof to the floor of the tunnel.

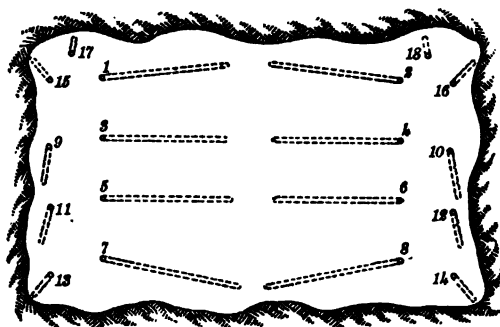


FIGURE 23.—Wedge-cut round of holes.

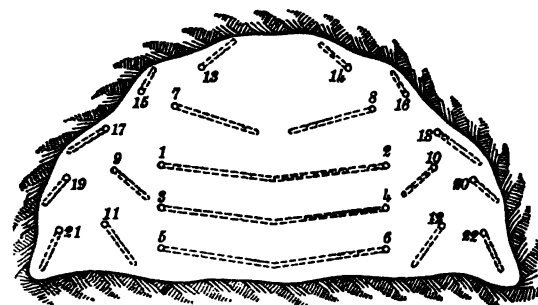


FIGURE 24.—Modified wedge-cut round for arched heading.

Figure 23 shows a typical wedge-cut round similar to the one employed in driving the Buffalo Water Tunnel. Holes 1 to 8 comprise the cut and were blasted simultaneously by electricity, whereas 9 to 14 are the side holes, and were next fired together, and 15 to 18 are the back or dry holes and were exploded last. Such a round must necessarily be changed somewhat if the heading is arched or semicircular. Figure 24 illustrates such a round, similar to those used in driving the heading of the large siphons on the Catskill Aqueduct. In this case holes 1 to 6, comprising the cut, were blasted together, followed by holes 7 to 12, which are called relievers, and finally by 13 to 22, which are called trimming holes.

Either vertical columns, as used in the cuts just cited, or a horizontal bar may be used to mount the machines when drilling this type of round; but if the majority of the holes are

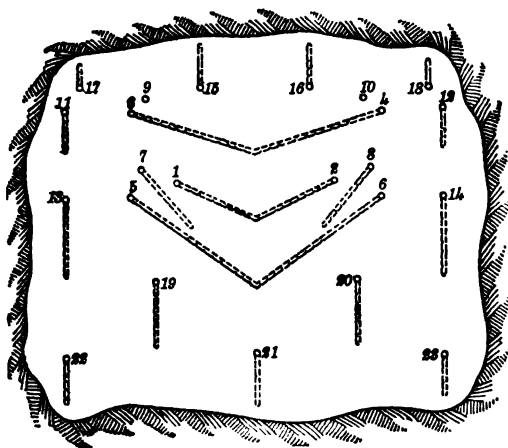


FIGURE 25.—Wedge-cut round drilled from a horizontal bar mounting.

to be drilled from one position of a horizontal bar, the position of the holes must necessarily be somewhat modified, although the general arrangement still remains a wedge-cut round. Figure 25 shows such an arrangement similar to the one employed at the Laramie-Poudre Tunnel. Holes 1 and 2 were called "short-cut holes," 3 to 6 "long cuts," 7, 8, 9, 10, 19, and 20 relievers, 11 to 14 sides, 15 to 18 backs, and 21 to 23 lifters, the numbering indicating the order of blasting. The lifters and two relievers (holes 19 and 20), which were used only in hard ground, were the only holes drilled from the lower position of the bar. Three machines were employed in drilling this round. The holes drilled by each and the order of drilling, the machines being lettered A, B, and C from left to right when facing the heading, are shown in the following:

Order of drilling for each machine at Laramie-Poudre Tunnel.

BAR IN UPPER POSITION.

Machine A.	Number of drill hole.	Machine B.	Number of drill hole.	Machine C.	Number of drill hole.
1	17	1	16	1	18
2	11	2	10	2	12
3	5	3	15	3	6
4	13	4	9	4	14
5	7	5	3	5	8
6	1			6	2

BAR IN LOWER POSITION.

7	19			7	20
8	22	6	21	8	23

A somewhat similar round was used with a horizontal bar at the Rawley Tunnel, and there are, of course, many other variations of the V-cut arrangement of holes, but the figures given illustrate the principles underlying the more common arrangements employed in tunnels and adits in this country.

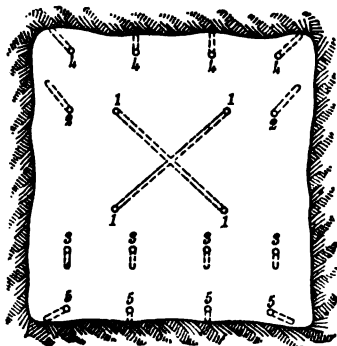


FIGURE 26.—Pyramid-cut round of holes.

The second general type of cut frequently employed may be designated as the pyramid cut, consisting usually of four cut holes drilled in such a manner that they meet, nor nearly meet, at or near a common point—generally near the axis of the tunnel—and when properly blasted remove a somewhat pyramidal core. Figure 26

shows a round of this type similar to the one employed at the Yak Tunnel in which the cut holes, No. 1, were blasted simultaneously,

followed by the remaining holes in the order indicated. In most of the instances observed by the authors the pyramid cut has been employed with vertical columns, but it can be drilled just as efficiently with the horizontal bar by drilling two or possibly three holes with each machine in the lower set-up. Figure 27 shows such a round.

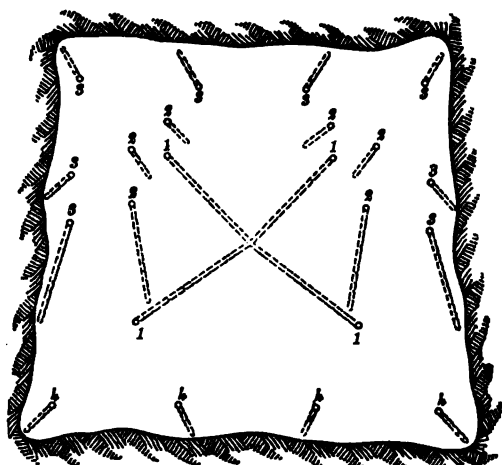


FIGURE 27.—Pyramid-cut round for use with horizontal-bar mounting.

The third type is the bottom or draw cut which was employed at several places visited, the one at the Carter Tunnel, illustrated in figure 28 being typical. The holes were blasted in the order indicated, Nos. 1 to 3 comprising the cut.

It can easily be proved theoretically that if a bore hole is drilled in a homogeneous mass of rock the maximum efficiency can be obtained from a suitable charge of explosive placed in it when the line of least resistance (by which is meant the shortest distance from the charge to a free surface of the rock) is at right angles to the axis of the bore hole; and that the minimum efficiency will be obtained when the two are coincident. Practically, also, although a homogeneous rock is a rarity, and hence the actual results will be influenced quantitatively somewhat by the various features of rock texture, such as joints, cracks, fault fissures, and bedding planes, the results have been found to agree in the main with the theoretical deductions. Obviously, therefore, in the heading of a tunnel or adit, where only one free face can be obtained, it is impos-

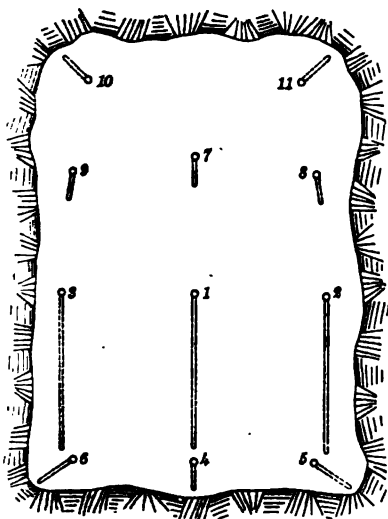


FIGURE 28.—Bottom-cut round of holes.

possible to drill and blast a single hole in such a manner that the maximum efficiency can be obtained from it. But by drilling a number of holes arranged according to any of the preceding systems, and blast-

ing the cut first so as to create more free surface much better results can be obtained from the holes that remain. It is for this reason that the position and direction of the holes comprising the cut are generally considered the most important feature of the work, the spacing of the remaining holes being admittedly merely a matter of having them sufficiently close together to break the rock into fragments of the required size for easy handling.

When the wedge or V cut is employed, the several pairs of holes should be placed close enough together for them to be of some mutual assistance, especially if the entire cut is exploded simultaneously. What this distance shall be is controlled almost entirely (as in the determination of the proper number of holes) by the character of the rock, its texture, toughness, the presence of cracks and bedding planes, etc. The separating distance is often determined by the foreman in charge, and if he is a man of wide experience, satisfactory results may follow; but the general efficiency of the work will often be increased greatly if experiments are made at the outset to determine just what combination will give the best results for the particular rock being encountered. It follows, of course, that such experiments should be repeated whenever a marked change in the nature of the rock is observed.

In order that the line of least resistance may approximate as closely as possible the perpendicular to the axis of the drill hole, the angle between opposite holes in the cut should be as large as can be obtained with any given depth of round. Hence, it follows that the drill holes should start as near as possible to the opposite sides of the heading; but obviously the full width of the heading can not be utilized because provision must be made for the feed screw and crank of the drill, which usually extend 3 to 4 feet from the face. This necessary allowance works especially to the disadvantage of narrow headings, because in them a greater proportion of the actual width must be sacrificed. But with broad headings the marked advantage of a wide angle is easily obtained and possibly offers an explanation of the popularity of the wedge-cut system in such headings.

Of even greater importance than the necessity of procuring a wide angle between opposite holes is that of drilling them so that they meet, or at least bottom near enough to one another to be detonated simultaneously by the one first to explode. Owing to mechanical reasons, the width of the drill bit, and hence the size of the holes, must be decreased with each successive change of steel, and as a result the hole is necessarily smallest at its bottom end—the place where the explosive is most needed and where it is extremely desirable that the holes should be as large as possible. Omitting from consideration the expedient of chambering—that is, the enlargement of the bottom of the hole by the explosion of a small primary charge

before loading it with the main charge of the explosive—which consumes entirely too much time to be considered for rapid tunnel driving, the defect can be overcome to a surprisingly large degree by the simple resort of connecting the drill holes, which concentrates the explosive at the point of the V. When fuse firing is employed, it is essential that the holes be so directed that they are intercommunicating (or so nearly so that both holes will detonate together) or the desired effect will not be gained, but when electric firing is employed direct connection, although very desirable, is not so absolutely essential.

In addition to the mere concentration of explosive thus obtained, the combined efficiency of the two charges is much greater than when exploded separately. If it be assumed that the holes are drilled in homogeneous rock and that they make equal angles with the shortest line from their junction to the free face, and if both are loaded with identical charges of explosive and detonated simultaneously, their maximum breaking effect will be exerted along the resultant of their combined forces, which in this theoretical case coincides with the shortest distance to the free face (the line of least resistance). Practically, of course, this result will be somewhat modified; but it is a well-established fact that if the ground is tough and difficult to break, much better results are obtained if the cut holes are directed and drilled to intersect—although, unfortunately, this is not widely known, as evidenced by the too great number of cases observed where no attempt was made to connect the cut holes.

Practically the same conditions prevail with the pyramid cut. The number of holes comprising it may vary from three to six, or even eight, according to the nature of the ground, and the proper number can best be determined by experiment. It is just as necessary to drill the holes with the widest possible angle between them, and it is even more essential that they meet in a common point, because one of the main advantages of this cut is the concentration of a greater quantity of explosive at the narrow apex of the core of rock to be removed. This advantage is lost if the charges of explosive in the different cut holes are not detonated simultaneously.

The bottom cut, as usually drilled in practice, although it often enables the attainment of a wider angle between the axis of the drill hole and the line of least resistance disregards entirely the important advantage to be obtained from connecting drill holes, and this, in the opinion of the authors, should be sufficient to prevent its use under any but exceptional conditions. For narrow mine adits, however, in which it would be impracticable, if, indeed, possible, to drill an effective wedge or a pyramid cut round, the bottom cut furnishes the only solution of the difficulty. In drilling in such a

tunnel it is recommended that the cut holes be drilled from as near the top of the heading as possible and directed in such a manner that they will connect with holes that are usually considered lifters and that both be detonated together.

DEPTH OF HOLES.

During the past four or five years there has been some difference of opinion among students of the problems of tunnel driving, as to the proper depth for drill holes in tunnel headings. In view of some of the remarkable results attained in driving the Simplon and Loetschberg Tunnels, where, as is agreed by everyone, the holes were much shallower than those in American practice, the question has been raised as to whether the holes in the tunnels of this country are drilled too deep. Numerous tables have been prepared in support of this argument, from which it appears that at most European tunnels the progress is much greater (in some cases more than twice greater) than that of tunnels in America. At the same time consideration is not always given the fact that in many instances the records are by nature in no wise comparable; for in Europe, at the majority of tunnels cited, the work was conducted throughout the entire 24 hours of each day, whereas in America in many instances only two shifts (and, indeed, in some only one) were employed daily. Then, again, the nature of the rock exerts an all-important influence upon progress, and in many cases this has been to the advantage of the European tunnels. A notable example of the influence of the rock encountered is found at the Loetschberg Tunnel, where the same methods and practically the same equipment were employed at the different ends, the north end working in limestone and the south end in gneiss and schist. The progress attained at the south end was much less than that at the north, in some months the progress in the north end being nearly double that in the south. Other considerations, also, especially the labor and the cost of driving, enter into the problem in such a manner as to make it impossible to say (when everything is taken into account) that the greater speed in European tunnels is due solely to the use of extremely shallow holes. That in many instances the holes in American tunnel headings are too deep, however, is impossible of denial, and hence a discussion of the factors that enter into the determination of the proper depth of holes is extremely desirable.

One of the chief advantages arising from the use of shallow rounds is (when the holes are properly directed) the increased efficiency obtainable from a given charge of explosive; for, as the width of the heading is for all practical purposes constant, the angle between the line of least resistance and the axis of the bore hole be-

comes a function of the depth of round, the width of the angle increasing with shallow holes. This advantage obtains especially with the wedge-cut and with the pyramid-cut, and it should be a fundamental consideration with the bottom-cut method of drilling the holes. Strangely enough, however, in the Loetschberg and the Simplon Tunnels, which are so often cited as examples of the "highly desirable" European practice of using shallow holes, this advantage was almost if not entirely thrown away because the holes were drilled in vertical rows and were nearly parallel to the bore of the tunnel. In such a case the line of least resistance and the axis of the bore hole are nearly coincident, a condition that results in the production of the least possible efficiency from the charge of explosive, and it can not be gainsaid even by the advocates of this method that a much greater quantity of explosive was required to break the same amount of rock than is usual in American practice. If to this is added the fact that such a system utterly ignores the advantage to be obtained from connected drill holes by the concentration of explosive at the apex of the core of rock to be removed, there is strong ground for rational suspicion that the extreme shallowness of the holes used in these tunnels was adopted from necessity rather than from desirability; with this system of drilling and directing the holes the difficulty of blasting out the rock with deeper rounds could not fail to be greatly increased.

Among other advantages of the use of reasonably shallow holes may be mentioned the fact that such a method allows the holes to be of larger diameter at their farther end, increasing their capacity for explosive and enabling its concentration at the point where it is most needed. This feature makes possible the European practice of employing extremely shallow holes, but it can hardly be denied that much more effective results in blasting might be accomplished by a change in the direction of the cut holes. Then, too, as in America at least, the holes are rarely charged with explosive to their full extent; the mass of rock between the ends of the charges of explosive in the different holes and the free face of the heading (which can be considered as a measure of the amount of resistance to be overcome) is not so great with the shallow holes. This fact or the customary use of relatively heavier charges in shallow holes may explain, perhaps, why in such cases the major part of the rock is usually thrown farther down the tunnel instead of being piled high immediately in front of the new face, with the double advantage of making loading of the rock easier and saving time in getting the drills mounted. It is fairly well established, also, that the rock tends to break into smaller fragments if shallow holes are employed. Again, if deep holes are not employed the same care in starting them exactly at a given point is not required, nor is it necessary to direct them

with such great accuracy, although of course the need of connecting the cut holes must not be overlooked.

The principal and unavoidable disadvantage in using the shallow hole round, on the other hand, is the fact that in order to obtain the same daily advance a proportionately greater number of drilling attacks must be made. This results in a waste of time in drilling, for it is possible under ordinary circumstances to drill one hole of a given depth more rapidly than it is two holes of the same aggregate footage, because of the time lost in changing to a new position, starting, etc. But even granting that the difference in drilling time (perhaps because it is too small or because in either case the drilling can be completed before the heading can be cleared of débris) is not an appreciable factor, each extra drilling attack required to obtain the same progress causes a corresponding loss of time in loading and blasting the holes, in waiting for the smoke and gases to be removed, in clearing the débris from immediately in front of the face, and in setting up the drills, all of which is ordinarily dead work and can not be avoided. This loss was seriously felt at the Loetschberg Tunnel, because in the endeavor to compensate for it four drills had to be employed in the heading (6 by 10 feet), and as a result the holes had to be drilled nearly straight, with disadvantages already described, because otherwise the drills in the center interfered seriously with the operation of those at the side.

On the other hand, if the holes are too deep the angle between the cut holes may be so narrow and the mass of rock in front of the charge of explosive may be so great that it will be impossible for the cuts to break bottom on the first blast, and thus the entire round is spoiled. The usual remedy in such cases is to blast the cuts separately and not to fire the remainder of the round until inspection has shown that the proper depth has been reached by the cut holes. Some delay can not be avoided when this method is employed, even if the holes break to the end, for it is never possible to return to the breast for such inspection immediately after the cuts have been detonated. But if the cut holes fail to break, the delay is greatly increased, because the remaining parts must be cleaned out, reloaded, and fired, with an additional delay in waiting for the smoke to clear.

This system was used at one of the Colorado tunnels, which at the time of first examination was being driven through some very tough rock. A round of holes slightly deeper than the average width of the heading was used, and it had given satisfactory results in the somewhat more frangible ground previously penetrated, the round being drilled and blasted in an 8-hour shift without difficulty, but when the harder rock was struck it became necessary to blast the cuts separately, and frequently to reload and shoot them for the

second and occasionally for the third time, the cycle being lengthened to about 10 hours, and several times at least 14 hours was needed. If three drilling shifts had been employed at the time, such a condition would have been fatal, but as only two attacks were being made the difference was not so noticeable, though even in this case the cost of the extra explosives required and the overtime wages of the men added a considerable expense to the tunnel work. Shortly after the first examination of this tunnel by the authors, however, the depth of the rounds was reduced to about 75 per cent of the width of the heading.

This made it unnecessary to load and shoot the cuts separately, and instead of getting two 7½-foot rounds in 20 to 22 hours, by working three 8-hour shifts it was possible to drill and blast four, and sometimes five, 5-foot rounds per day, thus increasing the daily tunnel progress from 15 to nearly 23 feet with only a small extra cost for labor. The consumption of explosive, a considerable item with the old system, was also decreased fully 25 per cent, and the total cost of the tunnel per foot was considerably reduced.

The disadvantage of too deep holes was strikingly brought out in the construction of the Laramie-Poudre Tunnel. During the first part of the work a 10-foot round was drilled in a heading 9½ feet wide, but the round was later changed to one of 7-foot depth with much better results. To be more specific, during the seven months from April 1, 1910, to October 31, 1910, at the east end of the tunnel, 3,171 feet was driven, an average of 453 feet per month, with a 10-foot round; but during the next 8½ months, from November 1, 1910, to July 24, 1911, when the tunnel holed through, 4,798 feet was driven, or an average of 545 feet, with a 7-foot round. This in an increase of over 20 per cent in spite of the fact that the greater speed was made when the work was at a greater distance from the portal; and, as there was no essential change in the methods or the equipment, or in the character of the rock penetrated, the increase is attributable solely to the use of shallower holes. When the 10-foot holes were employed to obtain an advancement of 8½ to 9 feet it was unusual to be able to drill and blast more than two rounds in 24 hours, and oftentimes not that many, as the average of 14½ feet daily testifies; but with the 7-foot round not only could three attacks be made, advancing on an average of 6.5 feet per attack, but a comfortable margin of time was left to provide for delays, and under favorable conditions this extra time meant extra footage. Thus, in March, 1911, the American hard-rock record of 653 feet, or over 21 feet per day, was established. This advantage of being able to complete an entire cycle of operations during a single shift should be given the weight it deserves in the problem. If crews of men could be found

who would work as well without rivalry and without special incentive to push the work, it might be perfectly feasible to choose a depth of round that would require 10 or even 12 hours for preparation, but under the present working conditions, where it is necessary to have some accurate measure of the work performed by each crew, a round is required for which the entire cycle can be completed during a single shift, with a sufficient margin of safety to provide for any ordinary delay.

It is, of course, impossible to set any definite standard or guide for the proper depth of hole that will be applicable to all cases. There are too many variables influencing the result. The proper depth must be determined by experiment in each individual case. However, from an extended examination of the results obtained from the methods employed in American practice, from a careful analysis of European practice as outlined in available published accounts, and from a study of all other procurable modern authority, the authors are of the opinion that for the majority of cases the proper depth of drill hole, the one that most equitably balances the advantages and disadvantages inseparable from the problem, is 60 to 80 per cent of the width of the tunnel heading. The following table gives an analysis of American practice in this respect:

TABLE 15.—*Depth of drill holes used in American tunnels.*

Name of tunnel.	Type of cut.	Height of heading.	Width of heading.	Average depth of cut holes.	Average depth of other holes.	Average depth of round drilled.	Percentage of width of heading of average round.	Character of rock penetrated.
Buffalo (water)	Wedge....	<i>Feet.</i> 8	<i>Feet.</i> 15	<i>Feet.</i> 8	<i>Feet.</i> 7	<i>Feet.</i> 7	46.5	Limestone.
Carter.....	Bottom....	7½	5½	9	8	8	106	Gneiss, granite, and porphyry.
Catskill Aqueduct:								
Rondout Siphon.	Wedge....	8	14	10	8	8	57	Limestone, sandstone, and shale.
Wallkill Siphon.	...do.....	8	14	12	10	10	71.5	Shale.
Moodna Siphon.	...do.....	8	14	10	8	8	57	Sandstone and shale.
Yonkers Siphon.	...do.....	8	14	8	6	6	48	Gneiss.
Central.....	...do.....	7	5	8	7	7	140	Do.
Fort William (water).	Bottom....	6½	5	6	5	5	77	Basalt.
Gold Links.....	...do.....	8	6	6	5	5	62.5	Granite and gneiss.
Gunnison.....	Wedge....	6	10	7	6	6	60	Altered granite.
Joker (drainage)	...do.....	11	12	10½	9	9	75	
Laramie-Poudre.	...do.....	6½	9½	8	7	7	74	Close-grained granite.
Lausanne.....	...do.....	8	12	8	7	7	58	Conglomerate, shale, and coal.
Lucania.....	...do.....	8	8	9	8	8	100	Hard granite.
Marshall-Russell.	Pyramid..	9	8	10	9	9	112	Granite and gneiss.
Mission.....	Bottom....	7	5	8	7	7	100	Shale and slate.

^a The height of the heading, instead of its width, is considered in this ratio when the bottom cut is employed.

TABLE 15.—*Depth of drill holes used in American tunnels—Continued.*

Name of tunnel.	Type of cut.	Height of heading.	Width of heading.	Average depth of cut holes.	Average depth of other holes.	Average depth of round drilled.	Percentage of width of heading of average round.	Character of rock penetrated.
		<i>Feet.</i>	<i>Feet.</i>	<i>Feet.</i>	<i>Feet.</i>	<i>Feet.</i>		
Mission (hard ground).	Pyramid.	7	5	8	7	7	140	Sandstone.
Newhouse.	do.	8	8	6½	5½	5½	69	Gneiss.
Nisqually.	Bottom.	11	9½	8	6½	6½	59	Rhyolite.
North west (water).	Wedge.	10	13	10	9	9	69	Sedimentary.
Ophelia.	do.	9	9	7	6	6	67	Granite.
Rawley.	do.	7	7½	9	8	8	106	Andesite.
Raymond.	do.	9	9	12	10	10	111	Gneiss and granite.
Roosevelt.	do.	6	10	7	6	6	60	Hard granite.
Swatch.	Bottom.	7½	6	5	5	5	67	Granite.
Snake Creek.	Wedge.	6½	9½	6½	5½	6	63	Dibase.
Spiral.	do.	10	16	12	10	10	63	Limestone.
Stilwell.	do.	7	7	6½	6	6	86	Conglomerate and andesite.
Strawberry.	do.	6	8	7	6	6	75	Limestone, sandstone, and shale.
Utah metals.	Bottom.	8	10	6½	6	6	75	Quartzite.
Yak.	Pyramid.	7	7	5	4	4	57	Limestone, sandstone, shale, and granite.

* The height of the heading, instead of its width, is considered in this ratio when the bottom cut is employed.

BLASTING.

SELECTION OF EXPLOSIVE.

To be suitable for use in tunnel work, as distinguished from surface blasting operations, an explosive should not produce any great quantity of poisonous gases and should not easily be affected by moisture. In common with other usages, a substance is required here that is stable in composition and not rapidly deteriorated by frequent changes in temperature or other causes; it must not be so sensitive to shock that safe transportation and handling is well-nigh impossible, as, for example, is the case with liquid nitroglycerin. Although under some circumstances, especially in tunnels that are not wet, an explosive called ammonia dynamite can be employed, the one that best fulfills the necessary requirements and the one that is almost universally used in tunnel work is known as gelatin dynamite.

Gelatin dynamite is a combination of a certain quantity of blasting gelatin (varying according to the strength desired) and a suitable absorbent. The former is made by adding a small percentage of guncotton (nitrocellulose) to liquid nitroglycerin, thus producing a jellylike mass that has greater explosive qualities than either of its constituents, but is much less sensitive to shock than nitroglycerin. The absorbent is usually some combustible material (wood pulp is frequently employed) to which has been added sufficient sodium nitrate to supply the necessary oxygen for its combustion. By the use

of such a combustible absorbent, instead of the inert one formerly employed with "straight" nitroglycerin dynamite, the gases generated by the burning of the wood pulp add to the volume produced by the detonation of the explosive constituent, and the extra heat generated in this combustion adds greatly to the total intensity of the reaction. Ammonia dynamites, which were discovered somewhat more recently, consist of a combination of ammonium nitrate and nitroglycerin absorbed in a so-called "dope" similar to that just described. The following tables^a show typical compositions of commercial samples of these two kinds of dynamite.^b

Typical compositions of commercial gelatin dynamite.

Ingredient.	Strength.						
	30 per cent.	35 per cent.	40 per cent.	50 per cent.	55 per cent.	60 per cent.	70 per cent.
Nitroglycerin.....	<i>Per cent.</i> 23.0	<i>Per cent.</i> 28.0	<i>Per cent.</i> 33.0	<i>Per cent.</i> 42.0	<i>Per cent.</i> 46.0	<i>Per cent.</i> 50.0	<i>Per cent.</i> 60.0
Nitrocellulose.....	.7	.9	1.0	1.5	1.7	1.9	2.4
Sodium nitrate.....	62.3	58.1	52.0	45.5	42.3	38.1	29.6
Combustible material ^a	13.0	12.0	13.0	10.0	9.0	9.0	7.0
Calcium carbonate.....	1.0	1.0	1.0	1.0	1.0	1.0	1.0

^a Wood pulp (used with 60 and 70 per cent strength), sulphur, flour, wood pulp, and sometimes resin used in other grades.

Typical compositions of commercial ammonia dynamite.

Ingredient.	Strength.				
	30 per cent.	35 per cent.	40 per cent.	50 per cent.	60 per cent.
Nitroglycerin.....	<i>Per cent.</i> 15	<i>Per cent.</i> 20	<i>Per cent.</i> 22	<i>Per cent.</i> 27	<i>Per cent.</i> 35
Ammonium nitrate.....	15	15	20	25	30
Sodium nitrate.....	51	48	42	36	24
Combustible material ^a	18	16	15	11	10
Calcium carbonate or zinc oxide.....	1	1	1	1	1

^a Wood pulp, flour, and sulphur.

The harmful gases usually resulting from dynamite are carbon dioxide and carbon monoxide. Although the former will not support respiration, and when present in sufficient amount may cause unconsciousness and even death from strangulation, it has no very injurious effects when sufficiently diluted. Carbon monoxide, how-

^a From a paper by Clarence Hall before the Am. Inst. Chem. Eng., Washington, D. C., meeting of Dec. 20, 1911.

^b For further discussion of the nature and composition of explosives, which is hardly within the province of this report, the reader is referred to the following bulletins published by the Bureau of Mines:

Bull. 15: Investigations of explosives used in coal mines, by Clarence Hall, W. O. Snelling, and S. P. Howell, with a chapter on the natural gas used at Pittsburgh, by G. A. Burrell, and an introduction by C. E. Munroe. 1911. 200 pp., 7 pls.

Bull. 17: A primer on explosives for coal miners, by C. E. Munroe and Clarence Hall, 61 pp., 10 pls. Reprint of U. S. Geol. Survey Bull. 423.

ever, is not only exceedingly dangerous, but its effects are also cumulative; indeed, if air containing even a small amount of it is breathed for any length of time, serious and often fatal results will follow. The fact that gelatin dynamite (and perhaps ammonia dynamite, which approaches it very closely in this respect) produces under proper conditions the least amount of carbon monoxide is one of its chief advantages for use in tunnel work. Even with this explosive, however, if the blasting cap is not strong enough to cause a complete detonation, especially if the dynamite burns rather than explodes, much greater amounts of carbon monoxide are formed; in addition there are many other harmful gases produced, among which may be mentioned the dangerous peroxide of nitrogen and hydrogen sulphide, the former being especially virulent.

The following table shows the results of tests conducted by the Bureau of Mines concerning the kind and amount of gases produced by the detonation of samples of various kinds of commercial dynamites. In making the tests a charge of 200 grams (approximately 7 ounces) in the original wrapper was exploded in a Bichel pressure gage, the gaseous products being retained and analyzed.

Gaseous products from explosives.

Kind of explosive.	Carbon dioxide.	Carbon monoxide.	Oxygen.	Hydrogen.	Methane.	Nitrogen.	Hydrogen sulphide.	Volume of gas.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Liters.</i>
40 per cent "straight" nitroglycerin dynamite.....	27.3	26.9	0.0	18.0	0.4	27.4		88.5
60 per cent "straight" nitroglycerin dynamite.....	22.2	34.6	.0	23.2	.8	19.2		128.9
40 per cent strength gelatin dynamite....	50.8	3.0	.0	1.8	.8	39.5	4.1	60.3
40 per cent strength ammonia dynamite.....	41.4	3.8	.0	3.1	.8	45.5	5.4	65.6
FFF black blasting powder (300 grams).....	49.7	10.8	.0	1.8	.6	28.4	8.7	67.8

A further distinctive feature of gelatin dynamite, winning for it the advantage over ammonia dynamite for most tunnel work, consists in its practically waterproof quality, a condition largely due to the insolubility of the blasting gelatin which can be freely immersed in water with little if any of it dissolving. Ammonia dynamite, on the other hand, being hygroscopic, has a great affinity for moisture, and hence not only can not be used in wet work (or even in damp work if the original paraffined paper covering has to be split), but greater care must be used in selecting a dry place for storing it.

Gelatin dynamite is somewhat less sensitive to direct shocks than other dynamites, and, unlike them, the sensitiveness does not increase with the strength; much stronger detonators must therefore be used,

even with the higher grades, in order to insure complete detonation. This fact is often not sufficiently appreciated by practical mining men, many of whom are not aware of the greater ultimate economy obtainable if the more powerful, although somewhat higher-priced, detonators are used with gelatin dynamites.

The strength of nitroglycerin dynamites as they are made to-day is generally rated according to the percentage of their nitroglycerin content in spite of the fact that both the volume of gases and the temperature (and hence the disruptive force) are augmented somewhat by the combustion of the absorbent material. Although 40 per cent is the strength most generally employed, they may be obtained in the following grades: 15, 17, 20, 25, 30, 33, 35, 40, 45, 50, 60, 70, 75, and 80 per cent. In the ammonia dynamites a part of the nitroglycerin is replaced by ammonia nitrate; but, as will be seen from the table on page 152, the rated strength of this dynamite is nearly the sum of the percentages of these two constituents. Ammonia dynamite is prepared in the same grades as nitroglycerin dynamite between 25 and 60 per cent. Owing to the strength of the blasting gelatin being greater than either of its constituents, the rated strength of gelatin dynamite is somewhat greater than the percentage of its explosive element. The usual grades of this dynamite correspond to those of nitroglycerin dynamite between 35 and 80 per cent, but it may also be procured in "100 per cent" strength.

The proper grade for use at any particular tunnel must be determined solely by local conditions. Such widely divergent results are obtained at different localities, although the same grade of explosive is used, and although the rock, as far as can be determined from its physical appearance and structure, is identical, that it is impossible to be dogmatic even with minute knowledge of local details. Generally speaking, however, a tough, close-grained, igneous rock will require a stronger explosive, whereas a sedimentary rock, or an igneous rock that has been altered and weathered, or perhaps shattered and broken, can be blasted just as effectively with a lower grade of dynamite. A notable example of the use of an extremely high-grade explosive is that of the Roosevelt Tunnel, where in the tough, close-grained, Pikes Peak granite "100 per cent" gelatin dynamite was required before satisfactory results were obtained. This is reported to have been the first "100 per cent" dynamite put to use in tunnel work. At the beginning of all work it is advisable to experiment with explosives of different strengths in order to determine which grade is best suited for the particular rock being penetrated, and it is, of course, obvious that similar experiments should be repeated whenever, owing perhaps to a change in the character of the rock, the dynamite being used fails to give satisfactory results.

The following table shows the grades of dynamite employed at the tunnels visited :

TABLE 16.—*Dynamite used at tunnels visited.*

Name of tunnel.	Kind.	Strength.	Remarks.
		<i>Per cent.</i>	
Carter.....	Gelatin.....	40	Some 80 per cent.
Catakill Aqueduct:			
Rondout Siphon.....	do.....	60	
Wallkill Siphon.....	do.....	60	
Moodna Siphon.....	do.....	75	
Central.....	do.....	40	
Gold Links.....	do.....	40	A small amount of 60 per cent.
Gumison.....	do.....	40 and 60	Mostly 60 per cent.
Laramie-Poudre.....	do.....	60	Some 100 per cent with the 60 per cent in cut holes.
Lanesme.....	do.....	60	
Los Angeles Aqueduct:			
Little Lake.....	do.....	40	Some 25 per cent and some 60 per cent.
Grapevine.....	Ammonia.....	40	Some 60 per cent and 75 per cent gelatin.
Elizabeth Lake.....	Gelatin.....	40	Tried 60 per cent and 70 per cent also.
Lucania.....	do.....	50	
Marshall-Russell.....	do.....	40 and 80	80 per cent also.
Mission.....	do.....	40 and 60	
Newhouse.....	do.....	40	100 per cent with 40 per cent in cut holes occasionally.
Niagualla.....	do.....	40	
Rawley.....	do.....	40 and 60	60 per cent in cut holes and lifters.
Raymond.....	do.....	40 and 60	
Roosevelt.....	do.....	40, 60, and 100	
Swatch.....	do.....	40	
Snake Creek.....	do.....	40	Some 35 per cent and some 60 per cent.
Stilwell.....	do.....	40	
Strawberry.....	do.....	40	
Utah Metals.....	do.....	40 and 80	
Yak.....	do.....	40	

The practice of loading the bottom part of the hole with 80 and even 100 per cent dynamite and using 40 or 60 per cent in the remainder is not now uncommon, especially in tunnels and adits in the Western States. It has the advantage of producing a greater disruptive force at the bottom of the hole, where such force is most needed, and at the same time it reduces somewhat the cost of explosives, especially as compared with an excessive amount of lower-grade dynamite. There is entailed, of course, the trouble of handling two different kinds of dynamite, not only in the heading but in the thawing house as well. Although in some tunnels where this procedure was tried the same results might possibly have been achieved by the use of shorter rounds, an alteration in the type of cut, or some other change in method, still the combined charge is useful, especially for exceedingly hard, tough rock.

It is obviously impossible to make any set rule for the determination of the proper quantity of explosive to be employed in tunnel work. There are entirely too many variable factors governed solely by local conditions. Various writers have derived from theoretical consideration formulas for the calculation of the proper charge of explosive for a blast hole, but the application of these rules is limited

to other types of blasting, such as quarrying or general mining, and they are not suited to the practical and actual conditions of tunnel work. For this the determination of the proper quantity of explosive is often left to the judgment of the foreman in charge, who, if he be widely experienced, can often produce excellent results; but the proper quantity can best be ascertained by a series of experiments in which the effects produced by different quantities of explosive are studied and compared.

It is very essential, however, that the charge of explosive be large enough. If it is too small and the cut holes fail to break bottom or the rest of the holes do not blast out their full share of rock it will be necessary to reload the parts remaining; this procedure not only requires fully as much explosive as if the holes had been properly charged in the first place, but also occasions a loss of time and footage, both of which are expensive. For this reason in a number of the tunnels visited it was customary to load the cut holes nearly to the collar. Although this is perhaps extreme as far as insuring that the cut holes break bottom is concerned, the extra dynamite helps to shatter the rock in finer fragments, thus making it easier for the shovelers to handle. Also, as no stemming^a is usually employed in such cases, a certain amount of the explosive probably acts in that capacity and increases the efficiency of the remainder of the charge. The very common practice of loading the lifters entirely full has a very different object—that of throwing the major part of the débris some distance away from the new face of the heading, thus making it easier for the drill men to get their machines at work promptly, and as the rock is scattered over a greater area the shovelers can attack it to better advantage. Such a practice is highly to be commended.

Data as to the exact amount of explosive actually employed in practice are difficult to obtain, chiefly because at many places an accurate record of powder consumption is not kept, but figures were obtained wherever possible at the tunnels visited. At the Gunnison Tunnel an average of nearly 30 pounds of 40 per cent and 60 pounds of 60 per cent gelatin dynamite was employed per round. This quantity is equivalent to approximately 5.5 pounds per cubic yard excavated. In driving the south heading of the Elizabeth Lake Tunnel the average for 1909 was 32.09 pounds^b of explosive per foot of tunnel, equivalent to 6 pounds per cubic yard. This figure, however, in-

^a In order to differentiate clearly between the material placed on a charge in a bore hole and the act of placing this material, the Bureau of Mines adopted the practice of designating the material as "stemming" and the act of placing it as "tamping." Readers should note this distinction which has been observed in the bureau's publications dealing with the testing or use of explosives.—Ed.

^b Aston, C. W., *The Elizabeth Lake Tunnel: Mine and Minerals*, September, 1910, p. 102.

cludes the dynamite used in trimming, so it is somewhat higher than the amount actually needed in driving. At the Rondout Siphon 175 to 200 pounds per round were required to drive an average of 10 feet^a with a heading approximately 120 square feet in area, equivalent to 3.9 to 4.5 pounds per cubic yard of rock excavated.

In advancing the heading of the Buffalo Water Tunnel, 3 pounds of 60 per cent dynamite were required per cubic yard.^b At the Laramie-Poudre Tunnel the powder consumption per cubic yard for March, 1911, was 3.9 pounds; for April, 4.7 pounds, and for May, 4.9 pounds. The average on the Little Lake division of the Los Angeles Aqueduct for May, 1911, was 4.5 pounds per cubic yard. At the Walkill Siphon Tunnel the average powder consumption per cubic yard ranged from 4.3 to 4.6 pounds. At the Yonkers Siphon Tunnel the powder consumption was approximately 4.5 pounds per cubic yard excavated.

The figures for the explosive used in the Simplon and the Loetschberg Tunnels indicate that the European average is somewhat higher than the American in this respect. At the Simplon Tunnel the charge was 6.5 pounds per cubic yard,^c and at the Loetschberg Tunnel the charge per round to obtain an average advance in the 6.5 by 10 foot heading of approximately 3.5 feet was 53 to 57 pounds,^d equivalent to 6.5 to 7 pounds per cubic yard.

The usual means of firing blasting charges, especially in tunnels and adits in the Western States, is by the use of a safety fuse. The term safety fuse originated from the fact that when properly used under working conditions this fuse burns at a uniform rate and does not flash or explode, as was often the case with the means employed for igniting blasting charges previous to its invention; but the term is somewhat misleading, because the fuse is not, nor has it ever seriously been claimed to be, safe for use in gaseous coal mines. The fuse used for tunnel work is composed of a core of gunpowder surrounded by various layers of waterproofing material.

Under ordinary conditions a safety fuse burns at a uniform rate, the variation rarely being greater than 10 per cent. In the European countries the normal rate is approximately 30 seconds per foot. In tests conducted by the Bureau of Mines with 14 samples of triple-tape fuse purchased for the Isthmian Canal the average rate of burning was determined as 26 seconds per foot for 3-foot lengths and 24.5 seconds per foot for 50-foot lengths. This rate is much faster than that of the fuse commonly employed in Western tunnel work,

^a Hogan, J. P., Progress on the Rondout Pressure Tunnel: Eng. Record, Jan. 1, 1910, p. 26.

^b Saunders, W. L., Rock tunnel records: Eng. Record, Aug. 27, 1910, p. 224.

^c Saunders, W. L., Tunnel driving in the Alps: Bull. Am. Inst. Min. Eng., July, 1911, p. 515.

^d Saunders, W. L., op. cit., pp. 538, 539.

40 to 45 seconds per foot being the customary rate of burning for fuse used there, although those figures are not the results of tests.

Experiments conducted by the Bureau of Mines^a prove conclusively, however, that the normal rate of burning of fuse is greatly changed by a number of conditions. Excess of pressure greatly accelerates it, and if the gases are sufficiently confined, the increase may be as great as 300 to 400 per cent. Although as great an increase as this would rarely be obtained in practice, the use of stemming that is too tightly packed or is impervious to the escaping gases may produce sufficient pressure to increase greatly the rate at which the fuse burns. When fuse is exposed to low temperature for a short time the rate is slightly increased, but if it is stored at tempera-

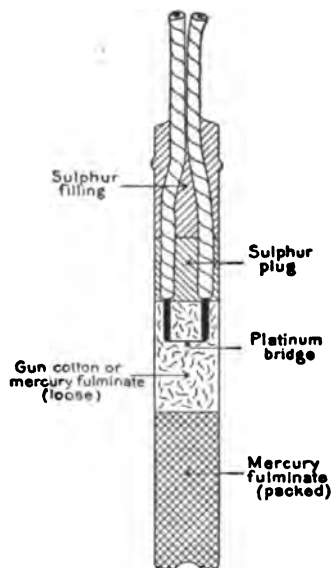


FIGURE 29.—Section of an electric detonator.

tures below freezing and handled before being warmed, cracks are apt to result in the waterproof composition which will permit the gas to escape and, by reducing the normal pressure, retard the speed of the fuse. Storage at high temperatures, however, causes a marked retardation which is apt to cause delayed shots and misfires; fuse should therefore never be stored near boilers or other places where the temperature is high. Moisture seriously impairs the efficiency of fuse, which should be carefully protected from it. Although the train of powder is covered with a waterproof covering throughout its length, the powder exposed at the end readily absorbs moisture and the cotton or hemp threads in the center act in the capacity of

sponges, so that the fuse for a foot or so from the end may be impregnated with moisture. When the fuse is lighted this water is driven ahead of the fire in the form of steam and delays the burning of the fuse, and if there is enough of it may become concentrated and extinguish the fuse entirely. Although twisting and bending seemingly have little effect upon the rate of burning, mechanical injury, such as pounding or crushing by falling rock, and abrasion, such as might result from the use of the tamping stick for consolidating the charge, greatly increase the rate. The results of the bureau's experiments show most conclusively that the greatest care should be

^a Snelling, W. O., and Cope, W. C., The rate of burning of fuse as influenced by temperature and pressure: Technical Paper 6, 1912, 28 pp.

taken in the storage and handling of fuse to prevent accidents from premature or delayed explosions.

A detonator or blasting cap consists of a copper cylinder closed at one end and about the diameter of an ordinary lead pencil, into which is packed some dry mercury fulminate and potassium chlorate. When used with a safety fuse, the end of the fuse is inserted in the open end of the copper cylinder, which is then crimped around the fuse by the use of suitable pliers. Under no conditions should anything but the proper tool be used for this purpose, because the fulminate of mercury is extremely sensitive to very slight shocks, and there is sufficient strength in a single detonator to produce disastrous results if discharged accidentally. In some tunnels, more especially those in the Eastern States, the detonators are ignited directly by an electric current. For this purpose special electric detonators are required in which the fuse is replaced by two suitably insulated copper wires joined at the inside end by a bridge of fine platinum or other high-resistance wire capable of becoming incandescent during the passage of an electric current. These are inserted in an ordinary detonator into which some guncotton has previously been placed. Caps prepared in this manner are called electric detonators and may be obtained from the manufacturers with wires of varying lengths as required.

Figure 29 shows the component parts of one of these detonators.

The strength of detonators is determined by the weight of mercury fulminate they contain and they are generally designated as triple X, quadruple X, etc.

The following table shows the weights of charge used in the different grades:

Weights of charges used in detonators and electric detonators of various grades.

Weights of detonator charges.		Weights of electric detonator charges.	
Commercial grade.	Weight of charge.	Commercial grade.	Weight of charge.
	<i>Grains.</i>		<i>Grains.</i>
3X or triple.....	8.3	Single strength.....	12.3
4X or quadruple.....	10.0	Double strength.....	15.4
5X or quintuple.....	12.3	Triple strength.....	23.1
6X or sextuple.....	15.4	Quadruple strength.....	30.9
7X or No. 20.....	23.1		
8X or No. 30.....	30.9		

The detonator chosen for blasting in tunnel work should be strong enough to produce complete detonation. With "straight" nitroglycerin dynamite, 3X caps were considered heavy enough, but with gelatin dynamites much stronger ones should be used, because this dynamite is not nearly as sensitive (being, of course, safer to handle).

The jellylike mass of the gelatin dynamite also has a tendency to retard the explosive wave as it passes along the bore hole and, therefore, requires a much stronger initial explosion to carry the wave with the same force through the entire length of the charge. The leading manufacturers all recommend nothing weaker than 6X caps for gelatin dynamite; although 5X caps have given results that were thought to be sufficiently satisfactory at some tunnels; at others, when a change was made to those of greater strength, the universal experience has been that the better results more than warranted the change, the common report being that "it pays." It is, therefore, here recommended that nothing weaker than 6X detonators (or double-strength electric detonators) be used with gelatin dynamite, which is practically the only kind employed in tunnel work. In addition to the less effective results produced by the lower-strength caps, the composition of the gases is greatly changed when detonation is not complete, and unsuspected and dangerous constituents may result. As we have seen, with complete detonations the gases are mainly carbon dioxide and nitrogen, with perhaps a small amount of carbon monoxide. With incomplete detonations a much greater percentage of dangerous carbon monoxide is formed from the nitroglycerin, and in addition the highly toxic peroxide of nitrogen is produced in larger or smaller amounts according to varying degrees of completeness of the reaction.

LOADING.

There has been much discussion lately regarding the proper position for the primer (as the particular cartridge of dynamite containing the detonator is called) when loading a blast hole, some arguing that it should be the last cartridge to be placed in position, whereas others claim the only proper place for it is at or near the bottom of the hole. One of the more common arguments for placing it at the top instead of the bottom of the hole is the fact that by so doing one removes the danger of igniting the dynamite from the side spitting of the fuse, which not only lessens the efficiency of the explosive, but produces dangerous gases. For it is obvious that when the fire in the fuse is compelled to travel past the full length of the charge, there is danger of the flame bursting through the waterproof covering and igniting the powder. However, an expert connected with one of the leading explosives-manufacturing companies states that gelatin dynamites (the kind generally used in tunneling) are less liable to be deflagrated in this manner than any of the others. Nevertheless, the objection, which is not applicable when electric detonators are used, is worthy of serious consideration when safety fuse is employed.

A second argument in favor of placing the cartridge last is the fact that the dynamite charge is much more apt to be packed firmly, thus eliminating air spaces which decrease the effectiveness of the explosive. For when the primer is the first or the second cartridge in the hole, the remaining cartridges and especially the one immediately following can not properly be pressed in place with safety, and air spaces are likely to be left. This objection, of course, applies equally to charges detonated by safety fuse or electricity.

A third argument is the fact that the detonation of a charge of dynamite in a bore hole takes place in a series of steps which follow each other with almost inconceivable rapidity but are nevertheless distinct. The first of these is the explosion of the cap, which in turn detonates the dynamite in the primer and causes what may be termed the primary explosion, which is in turn communicated to the remainder of the charge. Although, by employing a strong cap, the amount of dynamite detonated in the primary explosion can be greatly increased, it can never be large enough to disrupt the rock completely, the greater force of the secondary explosion being required for this purpose. If, then, the cartridge containing the detonator is placed at the bottom of the hole and fired, the primary explosion consumes the dynamite ineffectually at the bottom of the hole where the full force of the blast should be used; consequently there is much greater danger of losing the last 8 or 10 inches of the round, with a consequent decrease in daily advance. It is also claimed that in placing the primer at the bottom, the explosion of the detonator tends to force a certain amount of the charge from the hole; this contention, however, is debatable. But if the primer is placed at the top, the primary explosion does not consume dynamite that is essential, and the full strength is developed from the remaining part of the explosive, except in so far as it is influenced by the previously mentioned hindrance to the explosive wave caused by the use of gelatin dynamite. And again, by placing the primer at the top, during the primary explosion a certain amount of pressure is developed and the remainder of the charge is, therefore, detonated under that greater pressure, hence its effectiveness is increased. This applies particularly where stemming is employed, and it requires only 2 or 3 inches of clay stemming to produce the results. If clay stemming is not used an extra stick of dynamite placed on top of the primer, which acts partly in the same capacity, is of great assistance.

On the other hand, it is claimed that when the primer is placed on top of the charge the collar of the hole is apt to be knocked off by the explosion of a neighboring charge. There is some question as to whether this really would happen, but if it did it would be a

very strong indication either that the hole was misplaced or was too heavily loaded; for, as some one has said, "certainly the collar of a hole is no place for dynamite." And as to the objection that when the primer is placed at the top the fuse is liable to be torn out by flying rocks, the remedy is a very simple one—that of coiling the fuse carefully close up to the hole. And, finally, if the wave does not travel with enough force to the bottom of the hole, the matter can be remedied by the use of a strong detonator or by employing a higher grade of explosive, which would have the double effect of producing a greater primary explosion and of lessening the length of the charge, because less explosive would be required.

In view of the several considerations outlined above, it would appear that the primer should be placed at or very near the top of the charge. It is clearly recognized by the authors that, in some sections of the country at least, miners are accustomed to place the primer at the bottom. But it does not necessarily follow that a practice is correct because the miners so consider it, for many of them also think that dynamite exerts a greater influence downward than in any other direction.

The use of stemming in tunnel work is also a mooted question. When low explosives, such as blasing powder, having a slow rate of explosion are employed stemming is, of course, absolutely essential for confining the gases long enough for their full strength to become effective. But with dynamite, whose detonation is extremely rapid, almost instantaneous, many persons believe that stemming is not required, and this belief appears to be warranted by the results obtained without its use. A possible explanation for this is the fact that in tunnel work the holes are generally overloaded and hence the pressure produced by the extra few inches of dynamite charge tends to confine the gases generated by the remaining and effective part of the charge, which is, of course, also the function of stemming. Another probable reason is that the inertia of the column of air in the bore hole acts as an incomplete substitute for stemming of some more solid material. This can be demonstrated by the effect produced by exploding uncovered dynamite on some flat surface in the open, and this effect doubtless has given rise to the belief that dynamite exerts more force downward than in any other direction. It is unanimously agreed, however, by experts who have studied the subject that better results can be obtained from any properly loaded hole when more substantial stemming is employed. The quantity of stemming required depends, of course, upon the rate of detonation of the explosive. With black blasting powder it may be necessary to fill nearly all the remainder of the hole in order that the stemming may not be forced out before the reaction is complete and the full strength of the gases produced; but with ordinary charges

of gelatin dynamite 2 to 6 inches of well-packed clay stemming will in most cases be fully sufficient.

The use of stemming in tunnel work has several disadvantages. In the opinion of many, if indeed not a majority, of tunnel men they more than counterbalance any gain in efficiency of explosive from its use. In the first place it causes delay in loading the holes at a time when every minute is precious. Again, the majority of miners, and especially those in the Western States, are strongly biased against it, and anyone who has tried to overcome one of their prejudices will appreciate the difficulty that would be experienced in getting them to use stemming, although, of course, if stemming were absolutely essential to good results, mere prejudice on the part of anyone should not be allowed to stand in the way of its adoption; still, as this is not the case, the wishes of the miners are usually deferred to. But a more serious disadvantage of the use of clay or similar material for stemming is the danger attending its removal from a missed hole, one of the most prolific sources of accident in tunnel work. But if the stemming consists of an extra stick of dynamite, as is usual in tunnel work, the simple insertion of a primer on top of the unexploded charge is all that is needed to prepare the hole for refiring. For these reasons, then, although clay stemming is essential in a bore hole that is not overloaded, for tunnel work in which it is customary to use more rather than less "powder" than is required, it is not so necessary that clay, sand, or similar stemming be employed.

Another thing to be considered in connection with the loading of a blast hole is the necessity of having the dynamite properly thawed. Thawing is required not only because of due regard for the safety of the men (which alone should be more than sufficient), but also because frozen dynamite can not be properly packed in the hole, and air spaces can not, therefore, be avoided, and because frozen dynamite does not develop its full strength on detonation, so that there is a decided loss in effectiveness.

It is desirable that the cartridges have as nearly as possible the same diameter as the drill hole, and that they be slit (carefully, of course) along the side with a sharp knife just before they are placed in the hole. Slitting enables the explosive to conform to any irregularities in the shape of the hole. The position of the detonator in the priming cartridge deserves attention also. Experiments have shown that the maximum force from a detonator is developed in the direction of its length. For this reason the detonator should be inserted into the end of the cartridge and not obliquely in one side, as is often done in tunnel work. Nor should the fuse project into or be laced through the cartridge because of danger of setting fire to the cartridge instead of detonating it properly with the cap.

FIRING.

When the number of holes to be fired is large, the work of lighting the fuses is generally done by two men, but when there are only a few holes in the round one man is sufficient. If two men do this work, it is customary for each man to light the fuse of a corresponding hole (for example, opposite cut holes) at the same time, each calling out the hole as he lights it. The actual ignition of the fuse (which should be closely coiled and the free end split for one-half to three-fourths of an inch to expose the powder train) is accomplished in various ways. At some places a candle is used and at others an acetylene lamp. The much better practice, however, is to use what is called a "spitter"; that is, a short piece of fuse that has been slashed and partly severed at regular intervals of perhaps one-half inch so as to expose the powder train. When the end of a spitter is ignited, the fire travels along the spitter, and as it reaches one of the cuts it spits violently out of the side; if the spit is directed toward a fuse it is almost certain to cause proper ignition. Each fuse should be ignited separately, and no attempt should be made, as it sometimes is, to bunch several and ignite them all at once. When it is necessary to protect the ends of the fuse from water, an empty powder box may be employed.

In extremely wet tunnels it is sometimes necessary to use a fuse igniter. One form of igniter that has given good results at a number of places consists of a short cylinder of celluloid, closed at one end and having the same inside diameter as that of the outside of standard fuse, and containing a small quantity of gunpowder or similar explosive. In use the open end is slipped over the free end of the fuse, which, instead of being split, is cut square. The igniter fits the fuse tightly enough to be held in place by friction. When being lighted the igniter is, of course, protected from dropping water, and the celluloid is set on fire by a candle or other flame; as the igniter is unaffected by mere dampness, it burns until the powder charge is reached, when the flash seldom fails to start the fuse. These igniters are not expensive and are exceedingly useful in wet work.

When ordinary electric detonators are employed, the only operations required are those of connecting the wires and passing a current through them by closing an electric-light circuit, or by generating a current in a so-called "battery" which consists of a hand-operated magneto or dynamo. All the holes so connected are exploded simultaneously, the chief and most serious disadvantage of electric firing for tunnel work. As has been mentioned, blasting in tunnel headings to be effective must take place in several steps—the cuts first, followed by the relieves, backs, sides, and lifters. Therefore, with electric blasting, although it has the ad-

vantage of shooting the cuts simultaneously, it is necessary for some one to return to the heading and connect the wires leading to the charges in the holes to be fired in each of the succeeding steps; and as some time must always be allowed in order to permit the smoke to clear, and oftentimes no little shoveling is required to uncover wires that have been buried by a previous round, blasting a round in this manner takes much more time.

To overcome this defect, manufacturers of blasting supplies are trying to perfect a delay-action detonator which resembles an ordinary electric detonator except that the platinum bridge, instead of directly igniting the mercury fulminate, sets fire to a short train of gunpowder, so that an appreciable, although short time elapses before the flame reaches the detonating part. By making the powder train of two different lengths two delays are obtained, so that if the special detonators be used in connection with a detonator not containing a powder train, the blasting can be performed in three stages from a single connection of the wires and from only one closure of an electric current. When these devices are used in tunnel work an instantaneous detonator is usually placed in the cut holes, a "first delay" in the relievers, and the "second delay" in the remaining holes. Unfortunately, the special detonators have not as yet been perfected for more than two delays, and this limitation has undoubtedly prevented their more extensive use in tunnel headings. For such blasting three stages hardly give satisfactory results, because a fourth stage is essential for the lifters, whose function is to throw the material broken by the other holes from the immediate front of the new face. Moreover, with the horizontal-bar mounting, a fifth step is also desirable in order to permit one lifter to go off after the others and throw the material away from that side of the tunnel where the capstan end of the bar is to be placed, so as to afford plenty of room for the jack bar and to permit it to be screwed tightly in place.

However, the delay-action electric detonators that are now found on the market permit the blasting to be conducted in almost any number of steps that may be required. Like the ordinary electric detonators, they consist of a platinum wire bridge inclosed in a metal cylinder by a waterproof composition. In the other end of the cylinder (which, instead of being closed and containing mercury fulminate, is left open) a short section of ordinary safety fuse is inserted, crimped in place, and the joint waterproofed. An ordinary blasting detonator is placed on the outer end of this piece of fuse and an electric current is passed through the wires leading to the platinum bridge, when the fuse takes fire and burns until it ignites the detonator. By making the pieces of fuse inserted in the blasting caps of different lengths, any desired number of delays may be

obtained from one connection of the wires and one closure of the electric current. This device, therefore, overcomes the one great disadvantage of electric firing.

Chief among the advantages of electric firing is the certainty of detonating all of the cut holes simultaneously. Although, of course, if two holes are connected they may explode as one, it is impossible to make several pairs of cut holes—in a wedge cut, for example—explode together when fuse firing is employed. There will always be enough variation in the rate of burning of the fuse to prevent it, no matter how exactly the lengths of the fuses are cut. But when the cut holes are detonated simultaneously, as can be done with electric firing, each can assist the other, with a resulting increased effectiveness from the explosive. It is, of course, true that holes fired on the first and second delays can not be made to detonate absolutely at the same instant, even with electric delay-action detonators; but this is not essential, as the work of the succeeding holes does not approximate that of the cuts.

Another advantage of electric firing is the absence of smoke and dangerous gases caused by the burning of fuse. A large percentage of these gases is carbon monoxide, as is shown by the following analysis of gases obtained from burning fuse.*

Analysis of gases produced by the burning of fuse.

[A. L. Hyde, analyst.]

	Per cent.
Hydrogen sulphide.....	0.8
Carbon dioxide.....	32.7
Oxygen.....	1.4
Carbon monoxide.....	23.4
Hydrocarbons.....	4.1
Nitrogen.....	23.8
Hydrogen.....	13.8
	100.0

STORING.

The place used for the storage of explosives should be substantially constructed, well ventilated, protected as much as possible from fire or lightning, and should be kept locked to prevent the entrance of children or other irresponsible persons. At mines or quarries the ideal magazine is, of course, one of cement or of brick, but at most tunnels, where the work is usually of a somewhat temporary character, the cost of such a building is not always justified; the dynamite is stored usually in a short drift in the side of the hill or

* Snelling, W. O., and Cope, W. C., The rate of burning of fuse as influenced by temperature and pressure: Technical Paper 6, Bureau of Mines, 1912, p. 8.

in a log house. If neither of these can be obtained, a frame house will answer the purpose, although of course not as well. When a house is used, it should always be covered with corrugated iron or some similar fireproof material and care should be taken to remove any small sticks and grass from immediately around it. If considerable quantities of explosive are to be stored, the magazine should be situated at some distance from the rest of the work, and in every case the powder should be kept far enough from the tunnel buildings to prevent serious damage to them, or to the persons working in them, in the event of an accidental explosion. Obviously, dynamite should not be stored near a dwelling.

More than one kind of explosive, as, for example, black blasting powder and dynamite, should not be stored together, but there is no particular objection to the storage of different grades of the same kind of dynamite in the same building, except the possibility of confusion that might result from such a practice. Detonators, electric detonators, and fuse should under no circumstances be stored in the same building with dynamite, nor should the operation of placing caps on safety fuse be conducted at or near the magazine or thaw house. Tools should never be permitted inside of the magazine, nor should the boxes of dynamite be opened there. The floor of the magazine should always be constructed of wood, and it should always be kept free from grit and dirt.

THAWING.

Most dynamite freezes at a temperature of 45° to 50° F., and if it is subjected to such temperatures it must be thawed before it can be used. In tunnel work dynamite is generally thawed by spreading the sticks on shelves in a warm room or small building, separate and preferably somewhat removed from the main magazine. If the power for the tunnel work is derived from a steam plant, the waste steam is very often used to heat the thaw house. In this case, however, it is very essential that the steam coils be boxed or screened in such a way that it will be impossible for a stick of dynamite to fall upon a steam pipe, as a serious explosion might result. Nor should the pipes be so placed that any nitroglycerin exuding from the cartridges can fall upon them. As most thaw houses are insulated from the cold by having double walls or high-banked earth against the sides, only a little heat is required to keep them warm; if electricity is available, a cluster of incandescent bulbs is often used for heating, with good results. At other tunnels a special heater was observed, which was composed of one or more coils of iron or other high-resistance wire stretched between insulators on a suitable framework, usually of wood. When a heater of this type is employed it must be

protected from the danger of a stick of dynamite lodging upon the wires, because they are generally hotter even than steam coils (a red glow being not uncommon), and hence there is much greater danger of explosion from this source.

At one of the tunnels visited an unprotected heater of this type was employed, and when comment was made upon the fact that there was no protection, reply was made that this condition was intentional. The reason given was that any person entering the thaw house was supposed to turn off the current by means of a switch provided for that purpose, and that the knowledge that the coil was not protected would make the men more careful to see that this was done. Such reasoning is all right as far as it goes, but it does not provide for the contingency of a stick of powder falling off the shelves when no one is in the building, nor is the thaw house safe for some little time, even after the wires have been disconnected, as they do not cool off instantly. Taken in connection with the fact that this particular thaw house was only a short distance from the tunnel portal, that it had no lock, and that all timber and other tunnel supplies had to be hauled past it, the situation should have been marked, in the language of insurance, "extra hazardous." That it did not occasion some accident during the period of its use is truly marvelous.

PRECAUTIONS.

Carelessness in the use of explosives is one of the prime causes of accidents in tunneling; it follows, therefore, that every care should be taken in their handling. Although the following list of precautions, compiled from a number of sources, is not claimed to be complete, it is given here in the hope that it may once more repeat some of the precautions to be observed in the handling and use not of dynamite alone but of the accessories of blasting as well.

HANDLING.

Don't forget the nature of explosives, but remember that with proper care they can be handled with comparative safety.

Don't smoke while handling explosives, and don't handle explosives near an open light.

Don't shoot into explosives with a rifle or pistol, either in or out of a magazine.

Don't attempt to manufacture any kind of explosive except under the supervision and direction of a trustworthy person who is skilled in the art. Many serious accidents, which have destroyed lives or inflicted injury on persons and property, have been caused by such attempts.

Don't carry blasting caps or electric detonators in the clothing.

Don't tap or otherwise investigate a blasting cap or electric detonator.

Don't attempt to take blasting caps from the box by inserting a wire, nail, or other sharp instrument.

Don't try to withdraw the wires from an electric detonator.

STORING.

Don't leave explosives in a wet or damp place. They should be kept in a suitable, dry place, under lock and key, and where children or irresponsible persons can not get at them.

Don't store dynamite cartridges on end, as this increases the danger of nitroglycerin leaking.

Don't store or handle explosives near a residence.

Don't open packages of explosives in a magazine.

Don't open dynamite boxes with a nail puller or powder cans with a pickax.

Don't store or transport detonators and explosives together.

Don't store fuse in a hot place, as this may dry it out so that uncoiling will break it.

Don't keep electric detonators, blasting machines, or blasting caps in a damp place.

Don't allow priming (the placing of a blasting cap or electric detonator in dynamite) to be done in a thawing house or magazine.

THAWING.

Don't use frozen or chilled explosives. Most dynamite freezes at a temperature between 45° and 50° F.

Don't thaw dynamite on heated stoves, rocks, sand, bricks, or metal, or in an oven, and don't thaw dynamite in front of, near, or over a steam boiler or fire of any kind.

Don't take dynamite into or near a blacksmith shop or near a forge.

Don't put dynamite on shelves or anything else directly over steam or hot-water pipes or other heated metal surface.

Don't cut or break a dynamite cartridge while it is frozen, and don't rub a cartridge of dynamite in the hands to complete thawing.

Don't heat a thaw house with pipes containing steam under pressure.

Don't place a "hot-water thawer" over a fire, and never put dynamite directly into hot water or allow it to come in contact with steam.

LOADING.

Don't allow thawed dynamite to remain exposed to low temperature before using it. If it freezes again before it is used it must be thawed again.

Don't fasten a blasting cap to the fuse with the teeth or by flattening it with a knife; use a cap crimper. The ordinary cap contains enough fulminate of mercury to blow a man's head or hand to pieces.

Don't "lace" fuse through dynamite cartridges. This practice is frequently responsible for the burning of the charge.

Don't explode a charge to chamber a hole and then immediately reload it, as the bore hole will be hot and the second charge may explode prematurely.

Don't force a primer into a bore hole.

Don't do tamping with iron or steel bars or tools. Use only a wooden tamping stick with no metal parts.

Don't handle fuse carelessly in cold weather, for when it is cold it is stiff and breaks easily.

Don't cut the fuse short to save time. It is dangerous economy.

Don't worry along with old broken leading wire or connecting wire. A new supply will not cost much and will pay for itself many times over.

FIRING.

Don't explode a charge before everyone is well beyond the danger line and protected from flying débris. Protect the supply of explosives also from this source of accident.

Don't hurry in seeking an explanation for the failure of a charge to explode.

Don't drill, bore, or pick out a charge that has failed to explode. Drill and charge another bore hole at least 2 feet from the missed one.

MUCKING.

NUMBER OF MEN.

The number of men in the crew that removes the rock broken in blasting exerts an important influence upon the speed at which the tunnel can be advanced. If the vertical column or the drill carriage is employed and the remainder of the work can not proceed until the heading is cleared, every minute saved at this work can be transmuted directly into progress; and although with the horizontal-bar system, nearly all of the mucking is done simultaneously with the drilling, and the heading can ordinarily be cleared by the time the drillers have finished the upper round of holes, still if the crew of shovelers is not large enough to accomplish this promptly, the delay is serious. In any event it is essential for rapid progress that the muck be removed as speedily as possible, as there are always a great number of little things to be done by the shovelers, even after the main work of loading the débris has been accomplished. Further,

it is obvious that the removal of the muck in the shortest space can be accomplished only by a nice adjustment to conditions and the employment of the exact number of laborers proper for the purpose; for it must be remembered that the space in the tunnel heading is most restricted, and if too many men attempt to work there simultaneously they will so seriously interfere with one another as to offset any possible gain from the employment of the extra men. On the other hand, if too few men are at work it will be impossible for them to remove the débris within the time allowable. Analysis of the most satisfactory practice at a number of tunnels shows that, under the conditions prevailing in the heading, a man shoveling requires $2\frac{1}{2}$ to 3 feet of floor space. That is, for a tunnel 10 feet wide not more than four shovelers should be used simultaneously, whereas not more than two men can work to advantage side by side in a 6-foot heading. In addition to these, however, it is desirable to have a man or two at work picking down the rock pile in front of the shovelers, loosening bowlders, assisting in the handling of the cars, or doing any of the many other things that make for speed in loading the muck. Accordingly, at tunnels 6 to 10 feet wide the proper number of men in the mucking crew ranges from three to eight.

At the Loetschberg and other of the European tunnels two sets of muckers were employed, one of which would rest while the other was engaged in loading the car. This course was thought to be conducive to greater speed, because the men could work much harder for the few minutes it took to load a car if they had an equal time to rest during the loading of the next one. There is little doubt that this is true or that the heading can be cleared sooner when such a method is used, but because of the higher cost of labor in this country, especially in the Western States, it is greatly to be questioned whether the gain would be sufficient to make such procedure profitable. At the Loetschberg Tunnel the shovelers receive a daily wage of 80 cents* as compared with the \$3 or \$3.50 for like work in the western part of the United States, so that a double crew of only five shovelers (similar to those of the Loetschberg Tunnel) would entail in this country an extra cost of \$15 to \$17.50 per shift, as compared with \$4 in Europe. As the advance per shift in America rarely exceeds 7.5 feet, the extra cost of the double-crew system would amount to at least \$2 per foot of tunnel driven. This would be justified only if it obviated a corresponding delay, although even in that event the question could properly be raised whether a change or adjustment in some other phase of the work was not the better solution.

* Saunders, W. L., Tunnel driving in the Alps: Bull. Am. Inst. Min. Eng., July, 1911, p. 532.

POSITIONS OF WORKING.

The advantage of giving the men a rest from the grind of steady shoveling can be obtained without the necessity of extra laborers by changing their positions regularly according to the system in use at several of the tunnels visited by the authors. At the Laramie-Poudre Tunnel, where one of the best examples of this method was observed, the six muckers worked according to the following cycle of operations: As soon as a car (1) was filled with waste two shovelers, who will be designated as A and B, took it at once to the rear, while two other shovelers, C and D, jumped to an empty car (2) near by (which had previously been thrown off the track on its side), set it upright on the track and pushed it into a position to be filled. In the meantime the remaining two, E and F, stopped picking down the rock pile, took the shovels left by A and B, and started at once to assist C and D in filling car 2. Another car (3) was then brought up by A and B as near as possible to the car (2) being filled, and was thrown off the track on its side in the position formerly occupied by the second car. A and B then picked down the rock pile for the other four during the remainder of the time consumed in loading car 2. When filled this car was removed by C and D, while E and F set up the third car and filled it with the assistance of A and B. The fourth empty car was meanwhile brought up by C and D, who then took their turn at picking down, and the cycle was completed when E and F took the third loaded car to the rear, returning with another empty, and then resumed their original position on the muck pile. It will be seen that by this method every man spent at least one-third of the time in tramming or picking down the rock pile, either operation being easier work than that of shoveling, and amounting virtually to a rest which, although perhaps not as complete as if no work at all had been done during that period, was still sufficient to relieve greatly the hard monotony of shoveling.

The regularity and mechanical exactness of procedure with this system is a still more important advantage. Each man soon learns precisely what is expected of him for each step of the operation and hence there is absolutely no confusion, no lost motion. There is rarely any occasion for the foreman to give an order to the men except under unusual circumstances, and in consequence he does not acquire the habit of shouting at the men constantly, an unfortunate phase of this work only too noticeable at some of the tunnels visited; nor, on the other hand, do the men form the time-wasteful habit of running to him for guidance in every minor contingency that arises. The statement that the little things make for success is not claimed as original, but it can nowhere apply better than in planning the utmost work attainable in the limited space of a tunnel heading; in-

deed, this seeming detail of eliminated friction and confusion warrants and deserves the most serious consideration.

In addition to advantages in organization, the speed attainable with this method leaves little, if, indeed, anything, to be desired. Cars of 16-cubic foot capacity were filled at the Laramie-Poudre Tunnel ordinarily in three or four minutes, and on one occasion (which, however, was somewhat exceptional, as the men realized that they were being timed) only one minute and thirty seconds was needed. At the Rawley Tunnel, where a similar system was used with only four muckers, 25 cars, having a capacity of 17 cubic feet, were filled in exactly 2 hours, and on a different shift 20 cars were loaded in 1 hour and 45 minutes. These figures are from an accurately timed record kept by one of the authors. The usual time required for mucking at the Rawley Tunnel is not far from the average of these figures (which include all ordinary delay incident to making the cars up into trains), a value somewhat less than six minutes per cubic yard of rock loaded. This does not suffer by comparison with the Loetschberg Tunnel, where five minutes was required to fill a cubic-meter car (35.5 cubic feet)* by a crew of 10 men, with an extra minute to remove it when full and replace it with an empty one.

HANDLING CARS.

The method of handling the tunnel cars is still another detail of consequence in the operation of mucking. One of the most common arrangements is to have them trammed from the face by hand to a siding or switch where they are made up into trains and hauled to the portal by whatever means are provided for that purpose. This system, however, possesses some disadvantages. The switch must be moved frequently at no little expense and trouble in order to keep pace with the tunnel advance, or else it will soon be so far from the face that it is practically worthless. There is considerable loss of time while the loaded cars are being removed and the empty ones are being brought to the face; even although every effort be made to reduce this lost time to the minimum, the switch can not well be located nearer than 100 feet from the face, whereas in practice 300 to 500 feet is more apt to be the actual distance. Moreover, the full car must usually be taken by hand the entire distance to the switch before the empty can pass it; when this system is employed heavy cars, almost without exception, are used (for reasons later shown), and on this account to move them any distance by hand entails heavy exertions on the part of the mucking crew.

* Saunders, W. L., Tunnel driving in the Alps: Bull. Am. Inst. Min. Eng., July, 1911, p. 535.

At some tunnels this difficulty was obviated by extending two tracks all the way to the face and loading cars on each one alternately. Even this arrangement is not entirely satisfactory, because it requires the extra labor and trouble of laying two tracks instead of one, which must be done after the tunnel is cleared of débris and before the new round is fired and is therefore very apt to cause a serious delay in the whole work, especially if the shovelers are a little late in clearing the heading. In addition, the need for keeping the switch as close as possible to the face is not so apparent with this method as with the first, and hence this most necessary feature is apt to be neglected. In that event much time will be wasted in the course of a shift by the men tramping the cars an extra distance.

At one tunnel the necessity for a double track was avoided by covering the entire floor of the heading with steel plates for about 30 or 40 feet back from the face. The cars to be loaded could easily be jumped from the track onto the first of these plates and rolled as near the rock pile as necessary, and when one car was full an empty one could be shunted around it without difficulty and placed in position for loading while the full one was being rolled back upon the track and tramped to the siding when the trains were made up. Such a method is simple and effective and, except for the work of moving the siding ahead, requires little extra labor; most of the plates are needed in any event for the men to shovel from, so that the work of adding one or two more would scarcely be noticed. This procedure is recommended if for some reason it is necessary to employ cars of large capacity.

But in most cases, as was mentioned in the section on haulage equipment, it is much better to use cars of smaller capacity; then the empty ones are tipped off the track to allow the full ones to pass and can be righted when needed and placed back easily by two men, thus avoiding all the complications and extra work arising from the use of a siding or switch. The smaller cars are also easier to load, for, as they do not occupy as much space in the tunnel heading, there is more room for the shovelers to work; also, the sides of the smaller cars being lower, each shovelful of rock does not have to be lifted as high in order to get it into the car, saving both time and energy. They are likewise easier to handle in case of a derailment, and as fewer men are required for tramping them out of the heading when full, a larger percentage of the time of the shoveling crew can be spent in the actual process of loading.

When the smaller cars are used, however, the work of handling them must be thoroughly systematized in order to prevent waste of time through avoidable delays. Although similar to that in use at a number of the tunnels examined, the system employed at the Rawley Tunnel was, perhaps, more carefully planned in all the details

than were any of the others. Upon arrival at the heading the empty cars were pulled as near as possible to the full cars waiting to be removed, which at this juncture ordinarily stood on the track some 75 or 100 feet from the face of the tunnel. The mule was then detached from the empty "trip" and used to pull the full cars back to the car being loaded, usually the last one of the previous empty trip; or if they had all been loaded the full cars were pulled back as near the face as possible. The empty cars were then hauled up to the full ones and tipped off the track on their sides out of the way. All of this work was performed by the mule driver alone, except when the shovelers had completed the loading of the empty cars, when they assisted wherever possible in order to expedite the work. After seeing that the cars of the full trip were properly coupled up, the driver then started with them for the dump and the muckers took the two empty cars nearest the portal, set them on the track, and trammed them to the face where one car was again tipped off on its side while the other was being loaded. Unless the mule driver was delayed in getting to the heading so that he did not arrive before all of the cars of the previous trip were filled, the operation of getting the loaded trip out of the heading and an empty car again in position to be loaded rarely occupied more than three to five minutes. The remainder of the cycle was similar to that just described for the Laramie-Poudre Tunnel, each full car being trammed a short distance beyond the last one of the empty trip, which was then taken up to the face and thrown on its side ready for use without delay when needed.

To recapitulate, the chief advantages of this system are: (1) It does away with a switch near the heading; (2) the cars do not have to be trammed any great distance by hand—only a little more than the length of one trip—and the distance is constant and does not vary with the tunnel advance; (3) the minimum time is consumed in getting the full car out of the way and replacing it with an empty one; and (4) little time is lost in making up the trains to be hauled to the dump. It can not, of course, be used satisfactorily unless the cars are small enough to be handled easily by two men, but this is a matter that can be provided for in purchasing the equipment, and, as has been shown, the smaller car has other advantages that make it desirable for tunnel work. A system similar to the one outlined, modified, of course, to fit local conditions, is highly recommended for future tunnel work.

STEEL PLATES.

The use of steel floor plates from which to shovel rock broken by the blast has become so general that mention of this feature of mucking should hardly be necessary. The authors were much sur-

prised, however, to find at one or two tunnels, where otherwise there was little left to be desired in the line of organization and equipment, that the muck was being shoveled without the use of plates. Even the most cursory study of the efforts of the men in pushing the shovels into the rock pile along the uneven surface of the bottom of the tunnel, and comparing the time required to load a car with results at other tunnels where steel sheets were in use, soon made it evident that much energy and time were being wasted needlessly.

At one tunnel the plates were not used because it was necessary to excavate the floor on a curve instead of making it flat, though even here plates could have been employed by leaving a part of the waste material in the bottom of the tunnel to form a flat surface upon which the sheets could have been placed. Upon inquiry at another tunnel the following reasons were given for their nonuse: (1) The muck was so sticky that it would not be any easier to shovel from the plates than from the rock pile; (2) it was impossible to prevent the sheets from becoming bent, twisted, and jumbled up with the muck when the holes in the bench were blasted; and (3) it was a great deal of trouble to lay them in position before blasting and to handle them during the work of mucking.

Although it must be admitted in all candor that the stickiness of the muck in this instance made it difficult to handle under any conditions, there is no real reason to suppose that shoveling from a plate would have been more arduous than from the pile—quite the reverse. The second objection is somewhat more serious for it is true, especially with heavy blasts, that the sheets are sometimes caught up and twisted by the explosion and occasionally hurled for considerable distances down the tunnel. If, however, the plates are properly covered with waste rock from the previous round before blasting, such occurrences are so extremely rare as to be negligible, and this is ordinarily the remedy for such difficulty. At the particular tunnel under discussion the trouble was somewhat different. It was being driven with a heading nearly square, which was followed at a distance of 8 to 10 feet by a bench 3 to 4 feet high. The holes in the bench were drilled from the same set-up as those in the heading, and they “looked” down and away from the heading and hence toward and slightly under the position that would have been occupied by the steel sheets. It is not surprising that with this arrangement much difficulty should have been experienced in keeping the plates down during the short time they were tried. Objection should not have been taken to the steel sheets but to the design of the tunnel itself, which was being driven considerably higher than it was wide but would have served every purpose required of it equally well if the dimensions had been reversed. Such a change would not only have obviated the sheet trouble but would

also have made driving easier in other respects. Aside from this, the tunnel was not high enough to warrant the removal of the material in two operations, as was being done at the time it was visited. Since then, however, the bench has been abandoned, all of the material being excavated at once, which has made practicable the mucking from steel plates.

The third criticism is entirely a question of economy and can best be met by inquiring as to whether it is not better for the muckers to spend 15 or 20 minutes while the drillers are loading the holes, and possibly as much more time during the mucking, in doing work that will save itself several times over. For it can not be denied that a man can work to much better advantage and handle more rock during a given time if he shovels from a smooth surface. When he is shoveling from the pile the shovel can rarely be pushed more than an inch or two without encountering a piece of rock too big to be shoved aside, and there must therefore be a distinct stop while the shovel, with any load upon it, is lifted clear of the obstruction, only, in all probability, to encounter another one almost immediately. It is not surprising, therefore, to find that the experience at a large majority of tunnels leads to the conclusion that steel plates for shoveling are among the chief economies.

TIMBERING.

MATERIALS.

If the rock through which a tunnel is driven does not possess sufficient strength and rigidity to carry the weight of the superincumbent mass, artificial supports for the roof, sides, and sometimes for the bottom, are necessary to prevent the rock from falling, crumbling, or squeezing into the excavation. These supports may be timber, brick, stone, metal, or concrete. Because of its cheapness and availability in many mining districts and the ease with which it can be cut to the required sizes and shapes and placed in position, timber is the support most generally employed; and even if masonry or concrete lining is required in the specifications of the completed tunnel, timber is almost always used as a temporary support until the more permanent material can take its place.

In most underground situations seasoned timber is preferable to green because it is better able to resist decay. The bark should invariably be removed from round logs, as the space between it and the wood affords an excellent breeding place for many forms of wood-destroying insects, and the bark itself collects moisture and thus encourages the growth of fungi, which are the chief wood-destroying agents. Round timbers when properly peeled and seasoned are more

durable than square timbers cut from a similar log of the same size and age, because the corners of the latter are especially liable to decay. In young and small timber, such as is generally used for mining work, the outer half of the log is usually sapwood containing starch, sugars, proteids, and other soluble organic compounds—the foods upon which decay-producing fungi thrive, which are practically wanting in the heartwood. If, in the process of squaring up, the attempt is made to obtain the largest possible square timber from a given log, the corners consist largely of this easily infected sapwood and are accordingly most liable to conditions bringing about quick rotting. It is not surprising, therefore, to find in moist underground workings where square timbers have been in place for three or four years that the corners of the timbers have decayed to such an extent that they can be pried off down to the heartwood with a miners' candlestick or any other sharp instrument. It is, of course, true that the outer part of a round log also consists of sapwood, but the exposed surface has not been injured or bruised by the saw.

Although round timbers deteriorate much more slowly than square ones, they are not as easily handled in the tunnel, are harder to align properly, and it is much more difficult to reinforce them by the ordinary false sets. If timber must be transported long distances, the greater weight of the round sticks (especially if the logs are truncated cones—or “churn shaped” as the miners say—instead of being nearly cylindrical), the freight costs become excessive and square timbers must be used. Under these conditions the saving in transportation charges will often pay for some type of preservative treatment to be applied to the timbers before they are placed underground.

The best method of checking the growth of fungi, and so increasing the durability of timber, is to poison the source of their food supply. Although there have been many processes invented for this purpose, most of those in use to-day depend upon the injection of either zinc chloride or creosote. The former can not be used advantageously in wet situations, however, for as it is soluble in water it is soon leached out, leaving the timber just as susceptible to attack as before. But if creosote is properly applied it can not be washed out, no matter how much water passes over the timber, and for this reason it is the preservative generally employed in mining and tunnel work. It is, however, somewhat more costly than zinc chloride and, as it is a liquid, the transportation charges are considerably higher than on zinc chloride, which can be shipped in bulk.

Creosote can be applied as a surface coating by painting with a brush or simple immersion in a tank, or the log can be more deeply impregnated with the preservative by one of the more complicated processes involving heat, pressure, or vacuum. Although painting is

the least efficient method, it has the advantage of cheapness, and if carefully done will give fairly satisfactory results. Although dipping or simple immersion results in little if any greater penetration of the preservative, it insures a more certain filling and coating of the cracks, checks, and other imperfections of the log and thereby affords better immunity from decay. After the necessary equipment has been obtained, it is also in most cases cheaper than painting, because it is more economical of labor, it being easier to run a number of sets of timber through a vat by some form of mechanical conveyor than to paint the same number by hand. For either method the timber must be fully dried and seasoned beforehand; otherwise cracks in the wood due to the evaporation of the moisture will break the protective covering, which is only a thin one at best, and thus give the fungi access to the interior of the stick. It is perhaps unnecessary to add that these, as well as any of the other treatments, should be given after the timbers have been cut to form, so that the ends and the mortise openings may be coated as well as the sides.

If the extra cost of a more thorough impregnation is warranted, the "Bethell" process is widely employed for timbers that are to be placed in wet situations. By this treatment the timber is for several hours given a bath of live steam at perhaps 20 pounds pressure, after which it is subjected to a vacuum for three or four hours more, when creosote, heated to a temperature of approximately 160° F., is applied under pressure until the desired amount of the preservative is forced into the wood. "Burnettizing" is a process practically identical with this, except that zinc chloride is used in place of creosote. There are also a number of methods less frequently employed that are designed to effect economy in the amount of chemical required and differ chiefly in the manner of their application. In one or two of them the interior of the timber is impregnated with the less expensive zinc chloride, which is in turn protected from the action of water by treating the outer zone with creosote. A more complete discussion of processes than can well be included in this report may be found in Forest Service Bulletins 78 and 107, "Wood Preservation in the United States," and "The Preservation of Mine Timbers."

For permanent tunnel linings brick or stone were formerly the chief materials employed, and most of the older tunnels, both in this country and abroad, are lined in this manner. Such linings are expensive, however, and require a higher class of labor to place them in position than does concrete, their modern substitute, nor do they afford the same imperviousness. Although metal beams and posts are employed advantageously as roof supports in the main entries and gangways of some coal mines, high cost prevents their use except for this or work of similar importance. The best modern material

for permanent linings is undoubtedly concrete, and although its employment in this work has thus far been restricted chiefly to railroad,

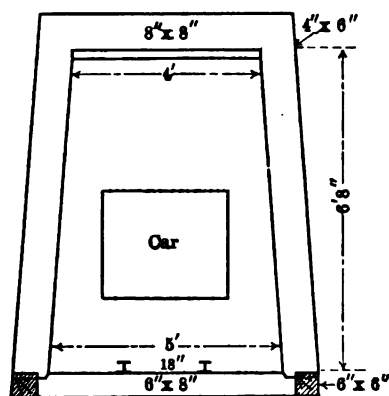


FIGURE 30.—Four-piece set for a small tunnel.

the weight of the posts a sill is placed for them to rest upon. Figure 30 illustrates such a four-piece set applied to as small a tunnel as it is usually advisable to excavate. The timbers are 8 by 8 inches, and instead of being partly beveled for withstanding side pressure, the posts are held apart by a 2 by 8 inch plank spiked to the cap. Figure 31 illustrates a common form of timbering designed for a tunnel of a convenient size for driving when a single track is all that is required. The timbers are 10 by 10 inches, and the joint between the cap and the posts is beveled at the corners so that the timbers can easily resist horizontal or vertical pressure without splitting. If heavier ground is encountered than can be held with

irrigation, or water-supply tunnels, its use in practically every important mining tunnel where a permanent lining is necessary must almost certainly follow.

TYPES.*

The simplest form of roof support is, of course, a single post, or a cap supported at either end by a "hitch" or recess in the side wall. If the sides of the tunnel are not strong enough to afford a hitch, the ends of the cap are supported by posts, and if the floor will not bear

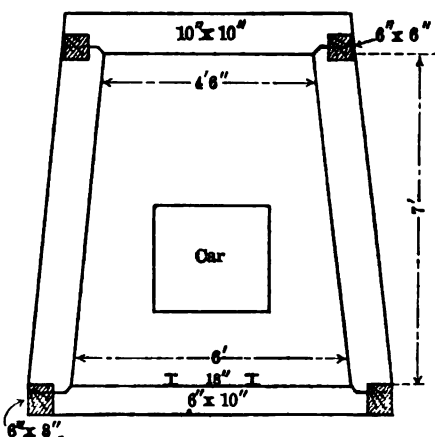


FIGURE 31.—Four-piece set for a medium-sized tunnel.

* As tunnels and adits for the purpose covered in this report are rarely too large to be driven as a single heading, the many complicated and ingenious systems of timbering that are used in driving large railway tunnels, either with multiple headings or single heading and bench work, need not be considered here. For a discussion of these methods the reader is referred to the monumental works of Drinker ("Tunneling, Explosive Compounds, and Rock Drills," by Henry S. Drinker, 1878, 1025 pp.), of Prelini ("Tunneling," by Charles Prelini, 1902, 307 pp.), and of Stauffer ("Modern Tunnel Practice," by D. Mc.N. Stauffer, 1906, 300 pp.), and to the publications of the civil and mining engineering societies.

this set, the posts and cap can be made of 12 by 12 inch timbers, with 8 by 10 inch sills and 8 by 8 inch braces or simply "braces," as they are commonly called.

In single-track tunnels where there is considerable traffic it is often advisable to have the opening wide enough to give room for a manway and the ventilating pipe on one side and the car tracks on the other, as shown in figure 32. Or if the tunnel has to carry a considerable volume of water the design shown in figure 33 has been used in many instances and has given excellent satisfaction, notwithstanding the fact that both the opening and the timbers are unsymmetrical. The 6 by 6 inch sill, which also forms the rail tie, is not notched into the post on the right side, but is merely held in place by a 2 by 10 inch plank spiked to the face of the post, the upper end of the plank being recessed for a depth of 4 inches to receive the sill. If even

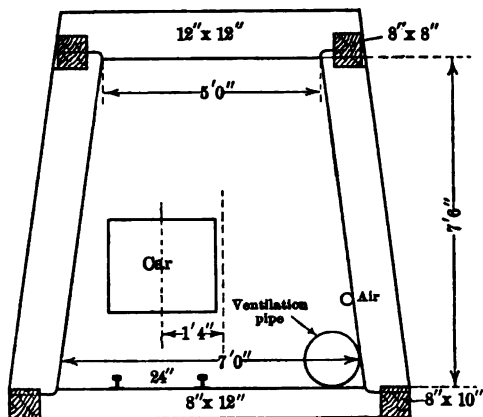


FIGURE 32.—Arrangement of timbering providing a manway at one side of the tunnel.

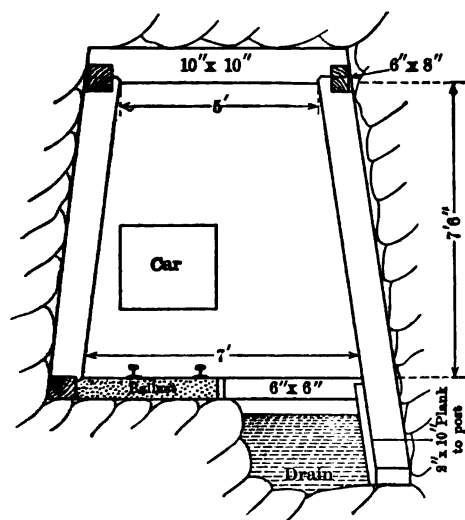


FIGURE 33.—Timbering for a wet tunnel.

Figure 35 shows the design of such a set for a tunnel 7 feet 6 inches high by 6 feet wide on the sills. The timbers are 8 inches square and the collar braces 4 by 6 inches. Instead of the braces being placed on the outer edges of the timbers, as is done on square sets where they can

larger volumes of water have to be provided for, the arrangement illustrated in figure 34 has given very good results at a number of places where it was tried. The amount of water and the grade of the tunnel of course determine the proper depth of the drain. The sill is supported by planks spiked to the posts, as in the preceding case.

Where the roof pressure becomes too great to be carried by a horizontal cap, what is known as the "arch set" is usually employed.

be slipped in from the outside, the collar braces on arch sets should be mortised into the face of the timbers in a central position, bisected by the joint, as shown in the illustration. By this means the level

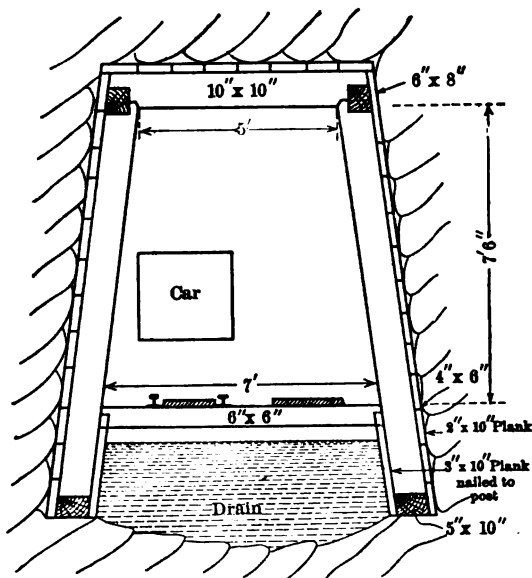


FIGURE 34.—Timbering for a tunnel producing a large volume of water.

pieces forming the arch, being much more difficult to hold in place while being blocked than are square sets, are prevented by the braces from slipping. If more room is required in the upper part of the tunnel than is given in this design, vertical posts are often used (see fig. 36), a substitution that not only increases the width of the tunnel at the shoulders, but calls for all timber cuts to be at an angle of 30

degrees, making the sides and the top pieces of the arch interchangeable. Figure 37 illustrates the arch system of timbering as employed in medium heavy ground for a tunnel 8 feet in width by 7 feet 6 inches in height, which is about the minimum size for a double-track tunnel. If the walls of a tunnel are sufficiently firm to stand without timbers and only the roof requires support, the arrangement shown in figure 38 can often be used to advantage. This system makes a carefully constructed footing for the arch timbers necessary, but hitch-cutting with a modern hand pneumatic drill is comparatively a cheap operation, the cost of which will be repaid many times by the saving in timbers and in the smaller quantity of rock to be excavated.

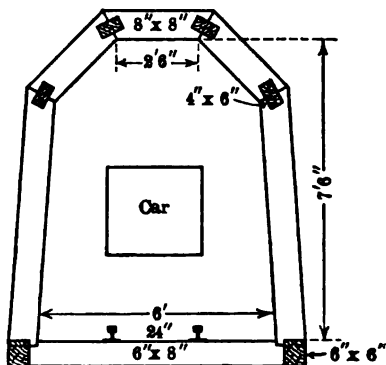


FIGURE 35.—Arch set for a small tunnel.

Swelling or creeping ground results from the exposure of certain rocks to the air, whereby they undergo chemical change and increase in volume so that the excavation not only closes in from the sides and roof but swells up from the floor as well. Under such conditions it may be necessary to design the timbers as shown in figure 39, the drain box and the track being protected by an inverted arch. If 12 by 12 inch timbers in this form will not resist the squeeze at the usual distance of 4 feet between centers, it is customary to close them up until they have sufficient resistance to withstand the pressure. Occasionally, however, zones of rock are encountered that can not be held even by this expedient, in which case the timbers can be kept from breaking by placing the sets about 3 or 4 inches apart; then whenever the pressure becomes too great it can be reduced by removing with a long-bladed pick whatever decomposed rock is in line with the open spaces, 4 or 5 inches back from the timbers. Swelling ground is usually so soft that this can be done without much trouble, and it is neither a difficult nor expensive matter to keep a wood-lined tunnel open until it can be

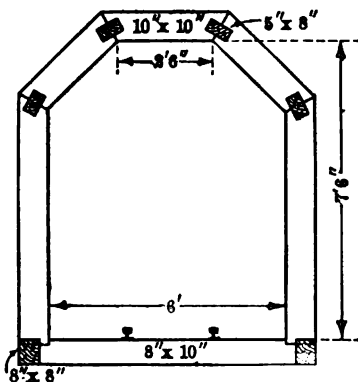


FIGURE 36.—Arch set with vertical posts.

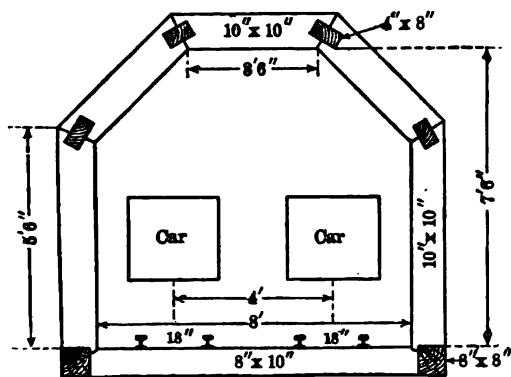


FIGURE 37.—Arch set for a double-track tunnel.

lined conveniently with concrete; which, by preventing access of air to the rock, will remove much of the difficulty. The octagonal set (fig. 40) offers another means of holding such heavy ground, and it is often supplemented with concrete in front of and between the timbers, as shown on the left side of the illustration. To insure the safety of the tunnel after the timbers have decayed, the sets should not be spaced less than 12 inches apart, and 15 to 18 inches is still safe as the wider opening gives room for a stronger rib of concrete between the timbers. The section is exceedingly easy to handle, and if the flow of water is not too great for the drainage area underneath, nothing better could be adopted. All the

pieces in this timber set are exact duplicates, which is a great convenience not only in framing but also in storage and erection.

An entirely different problem from swelling ground, and one temporarily much more difficult to handle, is often encountered where adits or tunnels have to be driven through shear zones, caved ground, or loose, crushed material that will not stand overhead without being supported as fast as it is opened. One of the oldest designs for driving through areas of this description is shown in figure 41, each alternate set carrying a double cap. This arrangement is simple, easily operated, and very satisfactory if the material in the roof or sides does not bring too much pressure

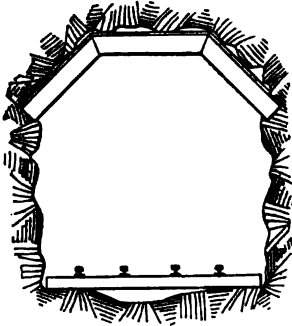


FIGURE 38.—Arch roof support and hitch.

on the spiling.^a The method possesses, however, two grave disadvantages—it requires two different sets of timbers and the spiling used must be long enough to cover both sets. The latter difficulty, if the overhead material is heavy, may prove serious, as it is often difficult to drive spiling across one space, to say nothing of two.

Under such conditions the tailblock system, illustrated in figure 42, is generally employed. As the timber sets are all the same size, it avoids one disadvantage of the preceding type, nor do the sets differ in any particular from those used with ordinary lagging. It offers, however, little improvement in the matter of driving the spiling in place. To be sure, the spiling does not have to cover two sets; but if the ground is heavy, the great pressure brought upon the tailblock by a comparatively small quantity of rock resting over the spiling as it is being driven forward creates an amount of friction that, added to the resistance in front of the spile, makes driving exceedingly

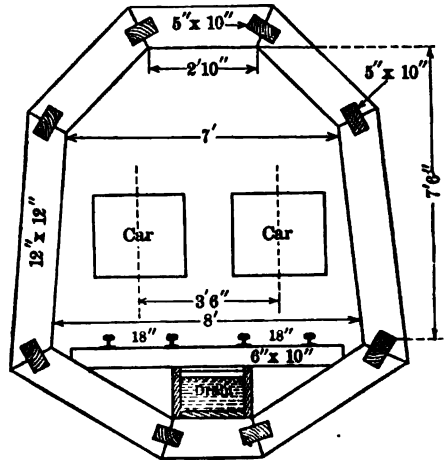


FIGURE 39.—Inverted arch set for swelling ground.

^a If the lining of a tunnel can easily be placed in position it is usually known as "lagging," but if it has to be sharpened to a chisel-shaped end and driven into position it is called "spiling" or "forepoling."

difficult even with the heaviest sledges that can be used. Although the greatest care be taken, the back end of the spile is often broomed and split by the heavy pounding required. This can be obviated in part by capping the back end of the spile with an iron shoe, and a heavy piston drill is sometimes employed to do the pounding. If the ground over the tunnel is very much shattered and weak, making the continued hammering on the spiling dangerous, it is safer to force the spiling slowly forward with light jackscrews.

When an opening has to be driven for any considerable distance through soft material requiring immediate support, the work can be expedited greatly by the use of what is known throughout the West as the swinging false set, illustrated in figure 43. Like many other inventions, this system was the child of necessity, and was first used in the Cowenhoven Tunnel, at Aspen, Colo., where the overlying rock brought so much pressure on the spiles that it was almost impossible to drive them forward with an 18-pound sledge. With this method no tailblocks are employed, nor need the spiling be driven across two sets of timbers, as in figure 41. The weight on the front end of the spiling is carried directly on

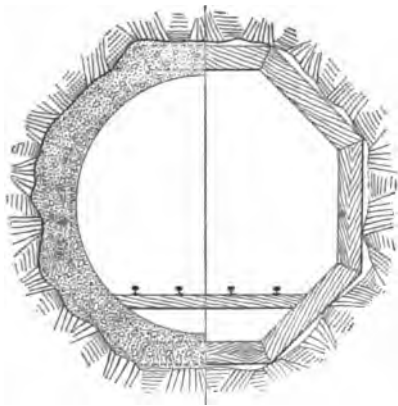


FIGURE 40.—Octagonal set for tunnel.

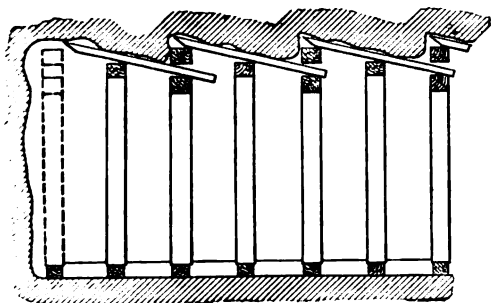


FIGURE 41.—Timbering for loose ground.

the position shown by the dotted lines. They carry a circular steel cap, *b*, which supports the front end of the spile *a*, so that the only pressure to be overcome in driving is that of the rock immediately above and in front of the spile, which can consequently be driven forward much more easily than in the tailblock system, in which a weight of 100 pounds on the front of the spile would easily cause a pressure of five times that amount on its supports. As the spiling is driven

the swinging false set, and the spiling can be driven into place with a quarter of the hammering necessary under the tailblock system. As will be seen by inspection of the longitudinal section, the posts of the swinging false set rest and rotate on the sill of the permanent set and when first erected occupy

forward the turnbuckle *c* is slowly unscrewed, allowing the swinging false set to fall forward and carry the point of the spiling in a nearly horizontal line. When all of the spiles have been driven home and the supporting block *d* has been placed under them, the turnbuckle *c* is unscrewed still farther, permitting the hanging rods to be unhooked from the eyebolts and the false set to be advanced to a new position, one set farther ahead. The system requires that the timbers for at least five or six sets from the face shall be bolted together in very much the same manner as hanging bolts are used in placing shaft timbers; this, however, is a direct advantage rather than otherwise, for by bolting the timbers together and screwing them tightly against the braces they can be placed in position much more easily and quickly. Further, the timbers are held together so firmly that if hard ground is encountered in any part of the face much heavier charges of explosives can be used than if they were held in place merely with blocks

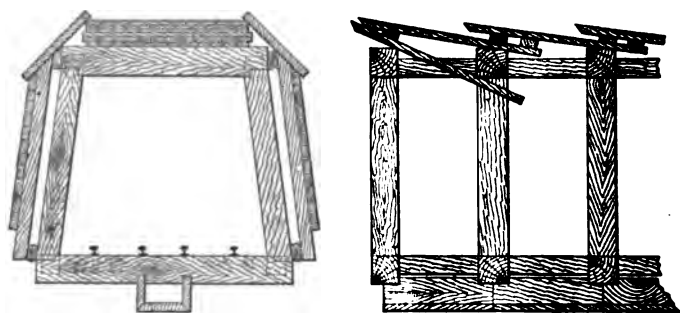


FIGURE 42.—Tailblock system of timbering.

and wedges. The swinging false set works equally well with square or arched sets, but if the latter are used the collar braces should be shifted from their normal position on the center line of the timbers, as shown in figures 36, 37, 39, and 40, to the outer end of the joints to make room for the greatest possible width of spiling; by this means the angle gap can be reduced to a minimum and the length of the "lacing," *e*, correspondingly reduced.

In driving a heading where the character of the rock necessitates timbering close to the face, care must be taken to thoroughly brace and block the front sets before firing. If the roof "breaks high" and there is any possibility of large masses dropping out of it, the space between the lagging and the roof must be completely filled either with waste or blocking; otherwise a large piece of rock may drop from the roof and pass completely through the lagging and thus endanger the lives of the men below. Timbering close to the face always diminishes the rate of progress by compelling the use of shallower holes and lighter charges. An excellent plan to admit

of heavier rounds under these conditions is to keep the last 6 or 8 sets of timbers firm and tight up against their collar braces by the use of tie bolts. These should be provided with center hooks to admit of their ready removal on the same plan as the hanging bolts that have so long been successfully used in shaft sinking. Only 6 or 8 sets of bolts are required, those from the rear being moved forward and used in the face. This system of tying the sets together has been found to be equally as advantageous in horizontal as in vertical driving.

The tunnel shield, such as is generally employed with or without compressed air for piercing subaqueous river-bed deposits, affords another solution of the problem of driving through soft ground. Although, in the matter of speed, safety, and economy, the modern shield leaves little to be desired, it has one great drawback, the initial cost of the installation, which practically bars it from use for

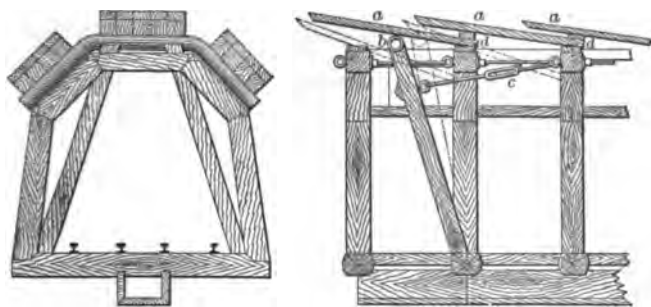


FIGURE 43.—Swinging false set for loose ground.

the narrow zones or small areas of running ground usually encountered by the class of tunnels considered in this report.

The system of timbering employed at the north end of the Elizabeth Lake Tunnel of the Los Angeles Aqueduct is of especial interest because of the ingenious and extremely effective means employed to drive through a comparatively hard rock which, however, was so shattered and broken that it could not be trusted to stand even temporarily without support; in addition, the speed and efficiency with which the timbering was placed in position were notable, enabling the north end of the tunnel to progress practically as fast as the south, although in the latter very little timbering was required. The following description is taken, with some condensation and rearrangement, from an article by Herrick:^a

The main tunnel, approximately 12 by 12 feet, was preceded by a short pilot heading approximately 8 by 8 feet, in which the roof

^a Herrick, R. L., *Tunneling on Los Angeles Aqueduct: Mines and Minerals*, vol. 31, October, 1910, pp. 135-143.

was supported by "false" timbering. Assuming for convenience the time of inspection to have been at the end of a shift, with the drills removed from the breast preparatory to blasting the round, the position of the timbers would have been somewhat similar to that shown in figure 44, *a*, which represents a short-timbered section of the full-sized tunnel and the horizontal timbers used temporarily in supporting the roof of the pilot heading. The posts of the permanent sets were 8 by 8 inch square timbers, 8 feet 6 inches in

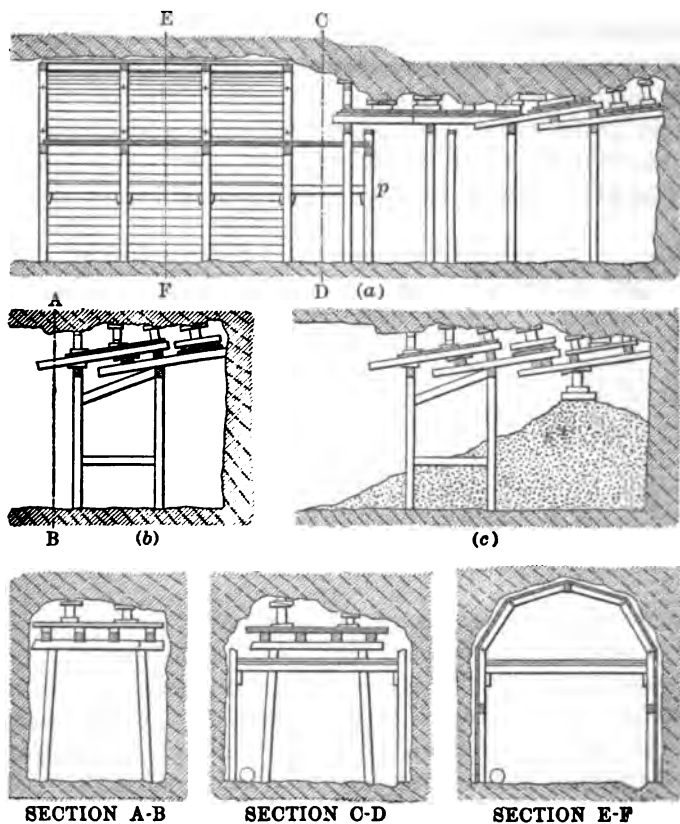


FIGURE 44.—System of timbering used at the Elizabeth Lake Tunnel.

length; and in the figure they are represented as spaced longitudinally at 8-foot intervals, although in practice they were often set irregularly, depending upon the weight of the roof. The width between the posts was 10 feet 2 inches in the clear, whereas the tunnel was broken as nearly as possible to a width of 12 feet. The collar braces were ordinarily 2 by 6 inch planks whose ends were supported either by wedges or timber ends spiked to the sets. The false posts in the heading were later used as permanent posts for the

full-size tunnel, and as one end of them was beveled to carry pieces of the permanent arch, the beveled ends were placed next the floor in the heading, whereas their squared ends supported the temporary caps, which likewise consisted of timbers already cut to form and later used as permanent posts. In this way there was no handling of heavy timbers not intended for permanent use. Just before blasting, the false timbers were carefully braced and wedged to the roof as tightly as possible, as shown in figure 44, *b*.

As soon as possible after the blasting, the timbermen went back to the heading to shore up the new roof temporarily from the top of the rock pile. For this purpose two horizontal timbers, supported from the broken rock close to the side walls and having transverse timbers and blocking resting upon them, as shown in figure 44, *c*, were placed by the timber crew, an operation that interfered little with the work of the muckers shoveling back from the face to allow the placing of the drills. The débris was next removed down to solid bottom to permit the setting of false posts, which, when capped, then carried the weight of the roof.

Timbering the tunnel during the enlargement to full size was not a difficult operation. Starting at the last permanent set and proceeding toward the face, new permanent posts (shown in figure 44, *a*) were placed in position as fast as the section was widened by picking down the side walls. Transverse spreader timbers, shown in view of section CD, figure 44, were then placed between these posts with their bottoms 14 inches below the joint and resting on timber ends spiked to the posts. Across these spreaders were laid two tiers of 2-inch plank, forming a floor 4 inches thick. The floor was some 2 or 3 inches below the bottoms of the caps resting on the false posts, so that it was easily laid while the false sets continued to hold the roof. Working from the end of this floor, the wedges and blocking transmitting the roof weight to the false set were next carefully knocked out and the shattered roof picked down on the floor, from which it was later shoveled into cars. By placing the permanent posts a foot or so in advance of the false posts, as in figure 44, *a*, the arch timbers of the permanent sets could be put in position as soon as the roof had been sufficiently removed. Lagging and wedging quickly followed, so that the roof was supported by the permanent sets shortly after the removal of the false blocking.

Although lining a mining tunnel with concrete is not, strictly speaking, a type of timbering, both have the same function, that of supporting the roof and walls. In this work the concrete is usually placed in the openings between the timbers and for a few inches in front of them, a disposition that, if the sets are not spaced too closely together, is generally sufficient even though later decay of the wood results in a corresponding weak spot in the lining. This defect can

be avoided, however, by the use of posts and caps made of reinforced concrete in place of wood, a practice that has been recently introduced and is finding great favor wherever its added expense is warranted. The concrete posts and caps are made outside of the tunnel in a mold that gives them the identical form of the wooden pieces they displace, and by proper reinforcement they can be made equal if not superior to timbers in strength, a strength that is practically permanent.

In water-supply tunnels a concrete lining performs the additional function of obviating eddies and friction against the otherwise irregular walls, and for this reason such tunnels are generally lined throughout, irrespective of the needs of the roof for support. On the Los Angeles Aqueduct the tunnel lining was generally at least 8 inches thick in places where the tunnel was not timbered, although

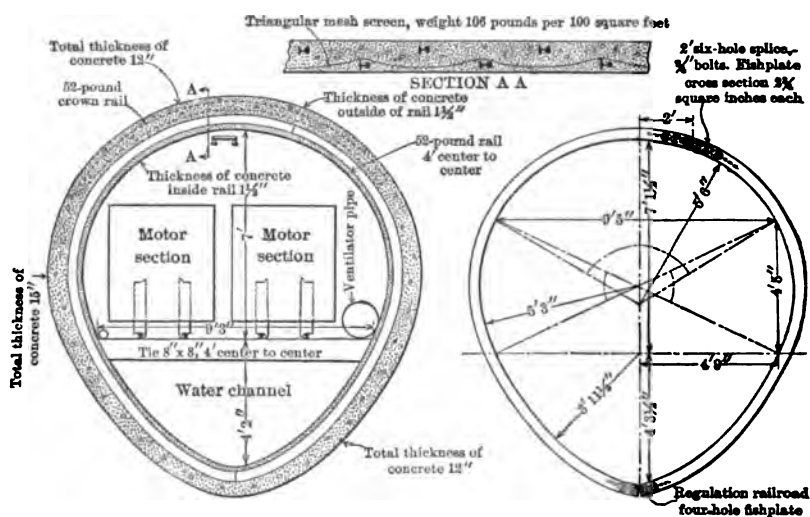


FIGURE 45.—Reinforced concrete lining at Snake Creek Tunnel.

an occasional rock projecting into the concrete was not removed unless it came within 4 inches of the inside finished surface of the lining. In timbered ground concrete was placed between the timbers and for a minimum distance of 4 inches in front of them. The inverted siphons of the Catskill Aqueduct were lined with concrete which was ordinarily 2 feet thick, but solid rock was permitted to project without removal to within 10 inches of the interior surface. Owing to the great hydrostatic head, in places as high as 700 feet, to which these linings were to be subjected, every piece of timber was removed before the concrete was put in place, and if it was necessary to support the roof during the time the concrete was setting, steel roof supports were designed and placed for this purpose.

At the Snake Creek Tunnel, where a zone of swelling ground was encountered which resisted all efforts to hold it in the ordinary way, the strongest timbers that could be obtained being crushed and broken in less than a month's time, a concrete lining reinforced with steel rails was installed. Figure 45, showing this lining, is practically self-explanatory.* If large volumes of water have to be carried through ground extremely difficult to hold, this design seems excellent, although its cost would be prohibitive for anything except important tunnels draining large areas of well-developed ground.

* A complete description may be found in the Engineering Record for May 25, 1912, p. 565.

COSTS OF TUNNELING.

In regard to publicity, the cost of tunneling is perhaps the most neglected feature of the work. Although the past 10 or 15 years have witnessed a very considerable amount of tunnel driving and there is presumably much information relative to cost extant, and although the articles describing methods, equipment, and other features of many of these tunnels have been numerous, only very meager data regarding the cost of the work, which is a very practical means by which the efficacy of methods and equipment can be measured, have found their way into the ordinary channels of publicity—the engineering periodicals. This possibly is due in part to the prejudice entertained by some contractors and tunnel men against a publication of their cost data; or they may actually not know what the work has cost them, aside, perhaps, from the difference between their bank account at the beginning and at the end of the job; others, perhaps, are unwilling to go to the trouble (for it does involve considerable extra labor) of preparing such matter for the magazines or other publications.

In an attempt to remedy this condition somewhat, there are set forth in the following pages as complete and accurate data as could be obtained, showing the cost of various phases of tunnel work at a number of different tunnels. Although the writers have not had the advantage of auditing the books from which these figures were taken and hence can not vouch personally for the absolute accuracy of the figures, the data were in all cases procured from persons in charge or who were in a position to know what the work actually cost. Accompanying the figures is a brief list of the more important features of the tunnel, without which it is impossible to make even an approximate comparison between any two pieces of tunnel work.

GUNNISON TUNNEL.

Important details.

Location: Montrose, Colo.

Purpose: Irrigation and reclamation.

Shape of cross section: Horseshoe.

Size: 10 feet wide at the bottom, 10 feet 6 inches wide at the spring line, 10 feet high at the spring line, 12 feet 4 inches high at the center of the arch.

Length: 30,645 feet.

Character of rock penetrated: Chiefly metamorphosed granite with some water-bearing clay and gravel, some hard black shale, and a zone of faulted and broken rock.

Type of power: Steam.

Ventilator: Pressure blower.

Size of ventilating pipe: 17 inches.

Drills: At first, pneumatic hammer, 4 drills in the heading; afterwards, pneumatic, piston, 4 drills in the heading.

Mounting of drills: Horizontal bar for the hammer drills, vertical columns for the piston drills.

Number of holes per round: 20 to 24 in the heading (approximately one-half of the tunnel).

Average depth of round: 6 to 7 feet.

Number of drillers and helpers per shift: 4 drillers and 2 helpers.

Number of drill shifts per day: 3.

Explosive: 60 per cent gelatin dynamite, with some 40 per cent.

Number of muckers per shift: 5 to 8.

Number of mucking shifts per day: 3.

Type of haulage: Electric.

Wages: Drillers, \$3.50 and \$4.00; helpers, \$3.00 and \$3.50; muckers, \$2.50 and \$3.00; blacksmiths, \$3.50 and \$4.00; motormen, \$3.00; brakemen, \$2.50 and \$3.00; power engineers, \$4.00.

Maximum progress in any calendar month: 449 feet.

Average monthly progress: 250 feet, approximately.

Cost of driving.

	Cost per foot of tunnel.
10,019 feet driven by undercut heading and subsequent enlargement.....	\$87.23
20,626 feet driven by top heading and bench.....	62.18
Average cost of excavation of entire tunnel.....	70.66

These costs include all labor, all materials, all repairs, all power, depreciation figured as 100 per cent on all equipment, with a proportionate charge for general (supervisory) and miscellaneous expenses of the entire reclamation project.

LARAMIE-POUDRE TUNNEL.

Important details.

Location: Home, Colo.

Purpose: Irrigation.

Cross section: Rectangular.

Size: 9½ feet wide by 7½ feet high.

Length: 11,306 feet.

Character of rock penetrated: Close-grained red and gray granite.

Type of power: Hydraulic at the east end, electric at the west.

Ventilator: Pressure blower.

Size of ventilating pipe: 14 and 15 inches.

Drills: 3 pneumatic hammer.

Mounting of drills: Horizontal bar.

Number of holes per round: 21 to 23.

Average depth of round: 10 feet at first, 7 to 8 feet later.

Number of drillers and helpers per shift: 3 drillers, 2 helpers.

Number of drill shifts per day: 3.

Explosive: 60 per cent gelatin dynamite, with some 100 per cent in the cut holes.

Number of muckers per shift: 6.

Number of mucking shifts per day: 3.

Type of haulage: Mules.

Wages: Drillers \$4.50, helpers \$4, muckers \$3.50, blacksmiths \$5, drivers \$4.50, dumpmen \$3.50.

Maximum progress in any calendar month: 653 feet, March, 1911.

Average monthly progress: 509 feet (for the 16 months when complete plant operated).

Special feature: Inaccessibility; the tunnel was located about 60 miles from the nearest railroad siding, and the roads were mountainous and very steep in places.

Cost of driving tunnel 11,306 feet.

	Cost per foot of tunnel.
Superintendents and foremen-----	\$1.50
Drilling-----	4.47
Mucking and loading-----	4.92
Tramming and dumping-----	4.63
Track and pipe-----	.47
Power house-----	.35
Blacksmithing-----	.84
Repairs-----	.47
Bonus to workmen-----	1.75
Maintenance of camps, buildings, and fuel-----	.62
Machinery repairs-----	.12
Air drills and parts-----	1.33
Picks, shovels, and steel-----	.84
Explosives-----	4.50
Lamps and candles-----	.42
Oil and waste-----	.38
Blacksmith supplies-----	.53
Liability insurance-----	.81
Office supplies, telephone, and bookkeeping-----	.86
	29.81
Permanent equipment (less approximately 10 per cent salvage)-----	9.73
	39.54

The permanent equipment included power plant, camp buildings and furnishings, pipes, rails, etc.

LOS ANGELES AQUEDUCT.

LITTLE LAKE DIVISION, TUNNELS 1 TO 10A.

Important details.

Location: Inyo County, Cal.

Purpose: Water supply, power, and irrigation.

Cross section: See figure 3.

Size: See figure 3.

Type of power: Electric power purchased at a nominal cost per kilowatt-hour from a hydraulic plant constructed and owned by the aqueduct.

Ventilators: Pressure blowers.

Size of ventilating pipe: 12 inches.

Drills: Pneumatic hammer, usually 2 in each heading.

Mounting of drills: Horizontal bar.

Number of holes per round: Usually 14 to 16.

Average depth of round: 6 to 10 feet.

Number of drillers and helpers per shift: 2 drillers and 2 helpers.

Number of drill shifts per day: Usually 1, but sometimes 2.

Explosive: 40 per cent gelatin dynamite, with some 20 per cent and some 60 per cent. Ammonia dynamite also tried.

Number of muckers per shift: Usually 5.

Number of mucking shifts per day: Usually 1, but 2 when 2 drill shifts were employed.

Type of haulage: Tunnels 1 to 3-N, mules; tunnels 3-S, to 10A-N, electric; tunnel 10A-S, mules.

Wages: Drillers and helpers \$3, muckers \$2.50, blacksmiths \$4, helpers \$2.50, motormen \$2.75, dumpmen \$2.50.

Cost of driving Tunnel 1B-S for 1,341 feet.

[Driven through medium-hard granite at an average speed of 225 feet per month.*]

	Cost per foot of tunnel.
Excavation	\$9.15
Engineering	.18
Adit proportion	.28
Permanent equipment (estimated)	2.35
Timbering (857 feet)	1.02
	12.98

In this tunnel, as in all of the tunnels of this division and of the Grapevine division, the cost of excavation includes the wages of shift foremen, drillers, helpers, muckers, motormen or mule drivers, dumpmen, blacksmiths and helpers, machinists, electricians (part), and power engineers; also the cost of powder, fuse, caps, candles, light globes, machine oil, blacksmith supplies and fuel, and machinists' supplies, and the cost of power and of repairs for power, haulage, compressor, and ventilating machinery.

"Engineering" includes the cost of giving line and grade, etc.

"Adit proportion" is a proportionate charge per foot of tunnel to defray the cost of an adit from the surface to the tunnel line.

"Permanent-equipment" costs were not segregated for each tunnel, but were compiled for the whole division, so the charge represents a proportionate charge per foot for the entire division cost, without salvage, of trolley and light lines, including freight and cost of installation; pressure air lines with freight and installation; ventilating lines with freight and installation; water lines with freight and installation; mine locomotives and cars, picks, shovels, drills and drill sharpeners, with repairs for the last four items.

Cost of driving Tunnel 2, length 1,739 feet.

[Driven through medium-hard but very wet granite at an average speed of 170 feet per month.]

	Cost per foot of tunnel.
Excavation	\$8.81
Engineering	.19
Adit proportion	.34
Permanent equipment	2.35
Timbering (1,590 feet)	3.28
	14.97

* The average speed given is computed on the basis of one heading per month.

Cost of driving Tunnel 2A, length 1,322 feet.

[Driven through medium-hard granite at an average speed of 150 feet per month.]

	Cost per foot of tunnel.
Excavation-----	\$8.05
Engineering-----	.18
Adit proportion-----	.34
Permanent equipment-----	2.35
Timbering (1,322 feet)-----	2.51
	18.41

Cost of driving Tunnel 3-N for 1,148 feet.

[Driven through medium-hard granite at an average speed of 150 feet per month.]

	Cost per foot of tunnel.
Excavation-----	\$10.10
Engineering-----	.23
Adit proportion-----	.51
Permanent equipment-----	2.35
Timbering (958 feet)-----	2.44
	15.63

Cost of driving Tunnel 3-S for 1,358 feet.

[Driven through granite of variable hardness, and containing pockets of carbon-dioxide gas, at an average speed of 155 feet per month.]

	Cost per foot of tunnel.
Excavation-----	\$12.38
Engineering-----	.28
Adit proportion-----	.16
Permanent equipment-----	2.35
Timbering (1,244 feet)-----	3.28
	18.45

Cost of driving Tunnel 3 (3-N and 3-S), complete, 4,044 feet.

[Driven through decomposed granite of medium hardness, dissected by slips and talcose planes requiring timber where ground was wet, and also containing pockets of carbon-dioxide gas, making work difficult and requiring extra provisions for ventilation. Average speed, 140 feet per month.]

	Cost per foot of tunnel.
Excavation-----	\$12.67
Engineering-----	.24
Adit proportion-----	.35
Permanent equipment-----	2.35
Timbering (3,570 feet)-----	2.71
	18.32

Cost of driving Tunnel 4, length 2,038 feet.

[Driven through medium hard to hard granite at an average speed of 145 feet per month.]

	Cost per foot of tunnel.
Excavation	\$12.00
Engineering	.24
Adit proportion	.16
Permanent equipment	2.35
Timbering (1,705 feet)	2.16
	17.01

Cost of driving Tunnel 5, length 1,178 feet.

[Driven through medium hard to very hard granite at an average speed of 120 feet per month.]

	Cost per foot of tunnel.
Excavation	\$11.10
Engineering	.21
Adit proportion	.08
Permanent equipment	2.35
Timbering (916 feet)	1.83
	15.57

Cost of driving Tunnel 7, length 3,596 feet.

[Driven through biotite granite of variable hardness at an average speed of 140 feet per month.]

	Cost per foot of tunnel.
Excavation	\$13.55
Engineering	.27
Adit proportion	.13
Permanent equipment	2.35
Timbering (2,609 feet)	3.60
	19.90

Cost of driving Tunnel 8-S for 1,334 feet.

[Driven through medium hard to hard granite at an average speed of 135 feet per month.]

	Cost per foot of tunnel.
Excavation	\$12.82
Engineering	.19
Adit proportion	.18
Permanent equipment	2.35
Timbering (126 feet)	.39
	15.98

Cost of driving Tunnel 9 for 3,506 feet.

[Driven through medium hard to hard granite at an average speed of 195 feet per month.]

	Cost per foot of tunnel.
Excavation	\$12.19
Engineering	.18
Adit proportion	.07
Permanent equipment	2.35
Timbering (305 feet)	.29
	15.08

Cost of driving Tunnel 10 for 5,857 feet.

[Driven through medium hard to hard granite at an average speed of 200 feet per month.]

	Cost per foot of tunnel.
Excavation	\$13.50
Engineering	.19
Permanent equipment	2.35
Timbering (194 feet)	.11
	16.15

Cost of driving Tunnel 10A-N for 1,496 feet.

[Driven through medium hard to hard granite at an average speed of 165 feet per month.]

	Cost per foot of tunnel.
Excavation	\$13.02
Engineering	.13
Permanent equipment	2.35
Timbering (24 feet)	.78
	16.28

Cost of driving Tunnel 10A-S for 2,200 feet.

[Driven through medium hard to hard granite at an average speed of 200 feet per month.]

	Cost per foot of tunnel.
Excavation	\$12.87
Engineering	.20
Permanent equipment	2.35
Timbering (215 feet)	1.15
	16.07

GRAPE VINE DIVISION, TUNNELS 12 TO 17B.

Important details.

Location: Kern County, Cal.

Purpose: Water supply, power, and irrigation.

Cross section: See figure 8, page 14.

Size: See figure 8, page 14.

Type of power: Electric power purchased from aqueduct plant.

Ventilators: Pressure blowers.

Size of ventilation pipe: 12 inches.

Drills: Pneumatic hammer, usually 2 in each heading.

Mounting of drills: Horizontal bar.

Number of holes per round: Usually 18 to 20.

Average depth of round: 6 to 8 feet.

Number of drillers and helpers per shift: 2 drillers and 2 helpers.

Number of drill shifts per day: Usually 2.

Explosive: 40 per cent ammonia dynamite, but 60 per cent and 75 per cent gelatin dynamite were employed in hard ground.

Number of muckers per shift: 4 or 5.

Number of mucking shifts per day: Usually 2.

Type of haulage: Electric after the first 400 to 500 feet.

Wages: Drillers and helpers \$3, muckers \$2.50, blacksmiths \$4, helpers \$2.50, motormen \$2.75, dumpmen \$2.50.

Cost of driving Tunnel 12, length 4,900 feet.

[Driven through hard granite at an average speed of 185 feet per month.]

	Cost per foot of tunnel.
Excavation	\$22.10
Engineering	.32
Permanent equipment	2.35
Timbering (90 feet)	.08
	24.75

Cost of driving Tunnel 13 for 1,525 feet.

[Driven through hard granite at an average speed of 130 feet per month.]

	Cost per foot of tunnel.
Excavation	\$20.60
Engineering	.10
Permanent equipment	2.25
Adit proportion	.37
	28.32

Cost of driving Tunnel 14, length 859 feet.

	Cost per foot of tunnel.
Excavation	\$22.70
Engineering	.13
Permanent equipment	2.25
Adit proportion	.72
Timbering (22 feet)	.16
	25.96

Cost of driving Tunnel 15, length 895 feet.

	Cost per foot of tunnel.
Excavation	\$23.28
Engineering	.11
Permanent equipment	2.25
Adit proportion	2.42
	28.06

Cost of driving Tunnel 16, length 2,723 feet.

[Driven through hard granite at an average speed of 145 feet per month.]

	Cost per foot of tunnel.
Excavation	\$20.07
Engineering	.17
Permanent equipment	2.25
Adit proportion	.55
Timbering (18 feet)	.04
	23.08

* These items include the same costs as for the Little Lake division. See p. 185.

Cost of driving Tunnel 17, length 3,024 feet.

	Cost per foot of tunnel.
Excavation.....	\$20.47
Engineering.....	.21
Permanent equipment.....	2.25
Timbering (142 feet).....	.22
	23.15

Cost of driving Tunnel 17½ for 1,345 feet.

[Driven through medium hard to hard granite at an average speed of 225 feet per month.]

	Cost per foot of tunnel.
Excavation.....	\$19.56
Engineering.....	.81
Permanent equipment.....	2.25
	22.12

Cost of driving Tunnel 17A for 3,275 feet.

	Cost per foot of tunnel.
Excavation.....	\$18.70
Engineering.....	.17
Permanent equipment.....	2.25
Timbering (441 feet).....	1.18
	22.30

Cost of driving Tunnel 17B for 4,915 feet.

	Cost per foot of tunnel.
Excavation.....	\$21.09
Engineering.....	.21
Permanent equipment.....	2.25
Timbering (168 feet).....	1.90
	25.45

ELIZABETH DIVISION, ELIZABETH LAKE TUNNEL.*Important details.*

Location: Los Angeles County, Cal.

Purpose: Water supply, power, and irrigation.

Cross section: Rectangular, with arched roof.

Size: 12 by 12 feet.

Length: 28,870 feet.

Type of power: Electric power purchased from aqueduct plant.

Ventilator: Pressure blower.

Size of ventilating pipe: 18 inches.

Drills: Pneumatic hammer, 3 in the south heading and 2 in the north.

Mounting of drills: Horizontal bar.

Number of holes per round: 25 in the south heading, 16 in the north heading.

Average depth of round: 8 to 10 feet.

Number of drillers and helpers per shift: 2 drillers and 2 helpers at the north end, 3 drillers and 3 helpers at the south end.

Number of drill shifts per day: 3.

Explosive: 40 per cent and 60 per cent gelatin dynamite.

Number of muckers per shift: 6.

Number of mucking shifts per day: 3.

Type of haulage: Electric.

Wages: Drillers and helpers \$3, muckers \$2.50, blacksmiths \$4, helpers \$2.50, motormen \$2.75, dump men \$2.50.

Maximum progress in any calendar month: 604 feet, April, 1910.

Average monthly progress per heading: 350 feet per month.

Cost of driving the north heading, Elizabeth Lake Tunnel.

[Driven through altered granite, requiring much timbering, 13,370 feet.]

	Cost per foot of tunnel.
Drilling and blasting-----	\$11.25
Mucking and tramping-----	11.70
Engineering and superintendence-----	1.27
Drainage-----	.45
Ventilation-----	.22
Light and power-----	5.55
Timbering (13,031 feet)-----	8.48
Cost of auxiliary shaft-----	.98
Permanent equipment (full charge, no salvage; estimated)-----	3.70
	43.55

Cost of driving the south heading, Elizabeth Lake Tunnel.

[Driven through medium hard to hard granite, requiring but little timbering, 13,500 feet.]

	Cost per foot of tunnel.
Drilling and blasting-----	\$14.65
Mucking and tramping-----	11.10
Engineering and superintendence-----	.86
Drainage-----	.17
Ventilation-----	.41
Light and power-----	4.98
Permanent equipment (without salvage; estimated)-----	3.70
Timbering (3,424 feet)-----	2.19
	38.01

LUCANIA TUNNEL.

Important details.

Location: Idaho Springs, Colo.

Purpose: Mine development and transportation.

Cross section: Square.

Size: 8 by 8 feet.

Length: 12,000 feet projected; 6,385 feet driven December 1, 1911.

Character of rock penetrated: Hard granite.

Type of power: Purchased electric current.

Ventilator: Pressure blower.

Size of ventilating pipe: 18 and 19 inches.

Drills: Pneumatic hammer, 3 in the heading.

Mounting of drills: Vertical columns.

Number of holes per round: 25.

Average depth of round: 8 to 9 feet.

Number of drillers and helpers per shift: 3 drillers and 2 helpers.

Number of drilling shifts per day: 1.

Explosive: 50 per cent gelatin dynamite.

Number of muckers per shift: 3.

Number of mucking shifts per day: 1.

Type of haulage: Horses.

Wages: Head driller \$5, drillers \$4, nipper \$3.50, boss mucker \$5, muckers \$4, drivers \$4, power engineers \$4, blacksmith \$5.

Maximum progress in any calendar month: 263 feet, September, 1911.

Average monthly progress: 125 feet per month for the first 4,800 feet, 240 feet per month for the last 1,575 feet.

Average cost of driving first 4,800 feet.

	Cost per foot of tunnel.
Labor	\$8.86
Powder	7.86
Fuse and caps	.17
Candles and oil	.21
Horse feed and shoeing	.18
Power	1.64
Repairs	.14
Tunnel equipment	2.75
Surface plant	1.25
	23.06

"Tunnel equipment" includes the cost of materials and installation of the pressure air line, the ventilating line, rails, ties and fittings, and the drainage ditch. "Surface plant" includes buildings, compressor, blower, transformers, motors, and drill sharpener.

Cost of driving next 1,575 feet.

The contractor received \$21.50 per foot to cover the cost of labor, powder, fuse, caps, candles, oil, horse feed and shoeing, power and repairs, and the installation of the tunnel equipment.

MARSHALL-RUSSELL TUNNEL.

Important details.

Location: Empire, Colo.

Purpose: Mine drainage, development, and transportation.

Cross section: Rectangular.

Size: 8 feet wide by 9 feet high.

Length: 11,000 feet projected; 6,700 feet driven January 1, 1918.

Character of rock penetrated: Granite and gneiss.

Type of power: Purchased electric current; also a small auxiliary hydraulic plant.

Ventilator: Fan.

Size of ventilating pipe: 12 and 13 inches.

Drills: 2, pneumatic hammer.

Mounting of drills: Vertical column.

Number of holes per round: 18 to 20.

Average depth of round: 9 to 10 feet.

Number of drillers and helpers per shift: 2 drillers and 2 helpers.

Number of drill shifts per day: 1.

Explosive: 40 per cent gelatin dynamite; with some, 80 per cent.

Number of muckers per shift: 4.

Number of mucking shifts per day: 1.

Type of haulage: Horses.

Wages: Drillers \$4, helpers \$3, blacksmiths \$4, helpers \$3, muckers \$3.25, trammers \$3.75, dumpmen \$3.25, power engineer \$3.50, shooters \$3.25.

Maximum progress for any calendar month: 187 feet, June, 1909.

Average monthly progress: 125 feet.

Cost of driving tunnel 6,700 feet.

	Cost per foot of tunnel.
Labor	\$9.37
Powder, fuse, caps, and blacksmith coal	3.35
Drills, steel, and repairs (less 30 per cent salvage)	1.34
Power	1.41
Permanent equipment and general expense (less 30 per cent salvage on permanent equipment)	8.41
	18.88

MISSION TUNNEL.

Important details.

Location: Santa Barbara, Cal.

Purpose: Water supply.

Cross section: Trapezoid.

Size: 6 feet wide at the base, 4½ feet wide at the top, 7 feet high.

Length: 19,500 feet.

Character of rock penetrated: Shale, slate, and hard sandstone.

Ventilator: Pressure blower.

Size of ventilating pipe: 10 inches.

Drills: 1 pneumatic hammer.

Mounting of drills: Horizontal bar.

Number of holes per round: 12 to 14.

Average depth of round: 7 to 8 feet.

Number of drillers and helpers per shift: 1.

Number of drilling shifts per day: 3.

Explosive: 40 per cent and 60 per cent gelatin dynamite.

Number of muckers per shift: 4.

Number of mucking shifts per day: 3.

Type of haulage: Electric.

Wages: Drillers \$3.50, helpers \$3, muckers \$2.75, blacksmiths \$4, helper \$3, motormen \$2.75, dumpmen \$2.50, power engineers \$2.75.

Maximum progress in any calendar month: 414 feet, February, 1911.

Average monthly progress: 210 feet.

Cost of driving the south portal, Mission Tunnel, May, 1909, to September, 1911, 5,515 feet.

	Cost per foot of tunnel.
Administration -----	\$1. 14
Labor -----	9. 20
Power -----	2. 12
Explosives -----	1. 97
Timbering (563 feet) -----	.30
Track and pipe -----	1. 22
Miscellaneous supplies -----	2. 46
Drill parts (including steel) -----	1. 02
Bonus -----	.48
	19. 91

"Administration" includes superintendence, office supplies, and general charges. "Miscellaneous supplies" includes candles, light globes, shovels, picks, blacksmiths' supplies and fuel, and machinists' supplies.

NEWHOUSE TUNNEL.

Important details.

Location: Idaho Springs, Colo.

Purpose: Drainage and transportation.

Cross section: Square.

Size: 8 by 8 feet.

Length: 22,000 feet.

Character of rock penetrated: Idaho Springs gneiss.

Type of power: Purchased electric current.

Ventilator: Pressure blower.

Size of ventilating pipe: 18 inches.

Drills: Pneumatic hammer.

Mounting of drills: Vertical column.

Number of holes per round: 14 to 22.

Number of drill shifts per day: 1 and 2.

Explosive: 40 per cent gelatin dynamite, with some 100 per cent in the cut holes.

Number of muckers per shift: 3.

Number of mucking shifts per day: 1 and 2.

Type of haulage: Electric.

Wages: Drillers \$4 to \$4.50, helpers \$3.25 to \$4, muckers \$3.50, motormen \$3.50, dumpmen \$3, blacksmiths \$3.50 to \$4.50, helpers \$3.

Cost of driving the Newhouse Tunnel.

	Jan. to Aug., 1909, 2,233 feet.	Sept. to Dec., 1909, 1,098 feet.	Apr. to Aug., 1910, 693 feet.
Labor -----	\$6. 72	\$6. 98	\$11. 73
Explosives -----	4. 15	3. 52	4. 57
Fuse and caps -----	.39	.36	.44
Transportation of materials broken -----	1. 49	1. 47	2. 22
Power -----	1. 90	2. 16	2. 82
Blacksmithing -----	1. 57	2. 61	2. 00
Use of drills, repairs and steel -----	1. 50	2. 74	2. 86
Equipment, ties, rails, pipe, etc. -----	1. 74	1. 78	2. 19
Sundries -----	.79	.80	1. 85
	20. 34	22. 42	30. 68

RAWLEY TUNNEL.*Important details.*

Location: Bonanza, Colo.

Purpose: Mine drainage and development.

Cross section: Trapezoidal.

Size: 8 feet wide at the base, 7 feet wide at the top, 7 feet high.

Length: 6,235 feet.

Character of rock penetrated: Tough, hard andesite.

Type of power: Steam with wood for fuel.

Ventilator: Pressure blower.

Size of ventilating pipe: 12 and 18 inches.

Drills: 2 pneumatic hammer.

Mounting of drills: Horizontal bar.

Number of holes per round: 23 to 25.

Average depth of round: 8 to 9 feet at first, 5 to 6 feet later.

Number of drillers and helpers per shift: 2 drillers and 2 helpers.

Number of drill shifts per day: 2 at first, 3 later.

Explosive: 40 per cent and 60 per cent gelatin dynamite (in the proportion of about 2 to 1).

Number of muckers per shift: 4.

Number of mucking shifts per day: 2 and 3.

Type of haulage: Horses and mules.

Wages: Drillers \$4.50, helpers \$3.75, muckers \$3.50, blacksmiths \$4.50, drivers \$3.50, power engineers \$4.

Maximum progress in any calendar month: 585 feet, July, 1912.

Average monthly progress: Approximately 350 feet.

*Cost of driving the tunnel 6,235 feet.**

	Cost per foot of tunnel.
Drilling and firing.....	\$5.25
Mucking.....	2.16
Tramming.....	1.13
Track and pipe.....	.44
Miscellaneous underground expenses.....	1.44
Power plant.....	2.50
Blacksmithing.....	.73
Miscellaneous surface work.....	.83
General expenses.....	1.98
Permanent plant.....	3.24
Timbering (1,618 feet).....	1.18
Boarding house, debit balance.....	.04
	20.98
Credit by salvage on permanent plant.....	1.11
	19.87

"Drilling and firing" includes labor, powder, fuse, caps, supplies, and repairs. "Mucking," "Tramming," and "Track and pipe" include labor and supplies. "Miscellaneous underground expenses" include wages of foremen, underground telephone, etc. "Power plant" includes labor, supplies, and

*A more detailed statement of the cost of this tunnel may be found in an article entitled "A problem in mining, together with some data on tunnel driving," by F. M. Simmons and E. Z. Burns, Bull. Am. Inst. Min. Eng., March, 1913, p. 369.

fuel. "Blacksmithing" and "Miscellaneous surface work" include labor and supplies. "General expenses" include salaries, office supplies, telephone, etc. "Permanent plant" includes machinery and buildings, with labor of installation, steel rails, permanent supplies, and repairs. "Timbering" includes labor and supplies. The salvage of the permanent plant is approximately 50 per cent on salable articles, such as machinery, rails, cars, etc.

ROOSEVELT TUNNEL.

Important details.

Location: Cripple Creek, Colo.

Purpose: Mine drainage.

Cross section: Rectangular, with large ditch at the side.

Size: 10 feet wide by 6 feet high.

Length: 15,700 feet.

Character of rock penetrated: Pikes Peak granite, chiefly.

Type of power: Purchased electric current.

Ventilator: Pressure blower.

Size of ventilating pipe: 16 and 17 inches.

Drills: 3 pneumatic hammer.

Mounting of drills: Horizontal bar.

Number of holes per round: 24, usually.

Average depth of round: 6 to 7 feet.

Number of drillers and helpers per shift: 3 drillers, 2 helpers.

Number of drill shifts per day: 3.

Explosive: 40 per cent, 60 per cent, and some 100 per cent gelatin dynamite.

Number of muckers per shift: 4, usually.

Number of mucking shifts per day: 3.

Type of haulage: Horses and mules.

Wages: Drillers, \$5; helpers, \$4; muckers, \$3.50; power engineer, \$4; blacksmith, \$5; helper, \$3.50; dump man, \$3.50; drivers, inside, \$5, outside, \$4.

Maximum progress in any calendar month: 435 feet, portal heading, January, 1909.

Average monthly progress: Portal heading, 300 feet; shaft headings, 270 feet; all headings, 285 feet.

Cost of driving tunnel.

Total cost of portal work.....	\$111,980.06
Contractor's percentage.....	11,404.88
Cost of shaft headings.....	262,128.55
Total cost of tunnel.....	386,421.49
Number of feet driven.....	14,167
Average cost per foot.....	\$27.27

Cost of driving the portal heading.

1908:	Feet.	Cost per foot.
February and March.....	514	\$22.690
April.....	262	30.970
May.....	268	26.760
June.....	187	35.010
July.....	208	29.600
August.....	300	21.760

1908—Continued.	Feet.	Cost per foot.
September	351	\$19.600
October	237	23.000
November	360	21.120
December	334	18.350
1909:		
January	435	16.410
February	290	22.206
March	340	21.745
April	316	21.266
May	402	18.762
June (8 days)	62	40.600

Cost of driving shaft headings.

1908:		
October (2 headings)	49	\$105.52
November (2 headings)	141	44.88
December (2 headings)	177	40.11
1909:		
January (2 headings)	261	24.06
February (2 headings)	601	23.70
March (2 headings)	639	26.256
April (2 headings)	670	25.02
May (2 headings)	552	28.34
June (2 headings)	498	27.375
July (1 heading)	319	32.871
August (1 heading)	410	27.747
September (1 heading)	355	32.40
October (1 heading)	380	28.178
November (1 heading)	298	34.20
December (1 heading)	251	35.153
1910:		
January (1 heading)	282	28.82
February (1 heading)	259	30.636
March (1 heading)	344	27.62
April (1 heading)	376	25.313
May (1 heading)	393	24.856
June (1 heading)	373	26.616
July (1 heading)	350	25.247
August (1 heading)	372	25.029
September (1 heading)	342	28.45
October (1 heading)	372	27.861
November (1 heading)	192	27.786

Typical distribution of expenses, portal heading, July, 1908, 203 feet.

	Cost per foot of tunnel.
Machinery and repairs	\$0.61
Air drills and parts	.96
Picks, shovels, and steel	1.90
Ditch men	1.09
Explosives	6.90
Candles	.36

	Cost per foot of tunnel.
Oil and waste.....	\$0.09
Electric power.....	2.08
Blacksmith supplies.....	.09
General expense.....	.18
Liability insurance.....	.17
Lumber, ties, and wedges.....	.01
Horses and feed.....	.01
Compressor men.....	1.79
Drillers and helpers.....	4.21
Blacksmiths and helpers.....	3.43
Muckers and drivers.....	4.11
Foremen.....	1.50
Bookkeeper.....	.12
	29.80

Typical distribution of expenses, shaft heading, February, 1910, 259 feet.

	Cost per foot of tunnel.
Maintenance of buildings, tents, etc.....	\$0.096
Machinery and repairs.....	1.158
Air drills and parts.....	1.930
Shovels, picks, and steel.....	1.930
Pipe and fittings.....	.193
Ditch men.....	1.490
Explosives.....	5.032
Lamps and candles.....	.217
Oil and waste.....	.252
Electric power.....	2.440
Blacksmith supplies.....	.150
Liability insurance.....	.213
General expense.....	.342
Lumber, ties, and wedges.....	.119
Horses and feed.....	.324
Machine men and helpers.....	4.050
Muckers.....	3.065
Blacksmiths and helpers.....	1.362
Engineers.....	1.300
Pipe and track men.....	.675
Drivers and dump men.....	2.355
Foremen.....	1.753
Mine telephone.....	.008
Bookkeeper.....	.193
	30.636

STILWELL TUNNEL.

Important details.

Location: Telluride, Colo.

Purpose: Mine drainage and development.

Cross section: Square, with ditch at side.

Size: 7 by 7 feet.

Length: 2,950 feet.
 Character of rock penetrated: Conglomerate and andesite.
 Type of power: Purchased electric current.
 Ventilator: Fan.
 Size of ventilating pipe: 10 inches.
 Drills: Started with electric drills, finished with pneumatic piston drills, using
 2 in the heading.
 Mounting of drills: Vertical columns.
 Number of holes per round: 18.
 Average depth of round: 6 to 6½ feet.
 Number of drillers and helpers per shift: 2 drillers and 2 helpers.
 Number of drill shifts per day: 1.
 Explosive: 40 per cent gelatin dynamite.
 Number of muckers per shift: 3.
 Number of mucking shifts per day: 1.
 Type of haulage: Horses.
 Wages: Drillers \$4.50, helpers \$4, muckers and trammers \$3.50, blacksmith \$4.50.
 Maximum progress in any calendar month: 170 feet, August, 1904.
 Average monthly progress: 150 feet (last 10 months).

Cost of driving the tunnel.

	Feet.	Cost per foot of tunnel.
1901-----	12	\$23.88
1901-2-----	490	22.98
1902-3-----	377	27.94
1903-4-----	702	21.69
1904-5-----	1,077	21.19
1905-----	292	30.37
Average for -----	2,950	23.88

These costs include all labor, supplies, repairs, powder, fuse, caps, candles, tools, lubricants, and general expenses, and the total value of the electric-drill plant with which the tunnel was started, and the total value of the air-drill plant which succeeded it, together with tunnel buildings, pipe, rails, and the ventilator, with no credit for salvage on any of this permanent equipment.

The fiscal year dated from September 30.

The tunnel was driven in 1901-3 with electric drills, and the high cost for 1905----- 292 30.37

STRAWBERRY TUNNEL.

Important details.

Location: Utah and Wasatch Counties, Utah.
 Purpose: Irrigation and reclamation.
 Cross section: Straight bottom and walls, with arched roof.
 Size: 8 feet wide by 9½ feet high.
 Length: 19,100 feet.
 Character of rock penetrated: Limestone with interbedded sandstone, and sandstone with interbedded shale.
 Type of power: Electric power generated in a hydraulic plant operated in connection with the tunnel. Distance of transmission from west portal to power house approximately 23 miles.

Ventilator: Pressure blower.
 Size of ventilating pipe: 14 inches.
 Drills: Piston pneumatic, usually 2 in the heading.
 Mounting of drills: Vertical columns.
 Number of holes per round: 16 to 18.
 Number of drillers and helpers per shift: 2 drillers and 2 helpers.
 Number of drill shifts per day: 3.
 Explosive: 40 per cent gelatin dynamite.
 Number of muckers per shift: 6.
 Number of mucking shifts per day: 3.
 Type of haulage: Electric after first 2,000 feet.
 Wages: Drillers \$3.50, helpers \$3.25, muckers \$2.75, motormen \$3.25, brakemen \$2.75; blacksmiths \$4, helpers \$2.75.
 Maximum progress in any calendar month: 500 feet, November, 1910.
 Average monthly progress: 320 feet per heading.

Cost of driving the tunnel.

West heading:	Feet.	Cost per foot of tunnel.
Previous to 1909.....	1, 613	\$60. 05
During 1909.....	3, 892	33. 58
During 1910.....	5, 021	30. 56
During 1911.....	3, 491	41. 52
January to July, 1912.....	2, 382	38. 79
East heading, October, 1911, to July, 1912.....	2, 682	33. 04
Average for.....	19, 081	38. 78

Detailed cost of driving the west heading for the year 1909, 3,892 feet.

Labor:	Cost per foot of tunnel.
Engineering.....	\$0. 49
Superintendence.....	. 73
Shift bosses.....	1. 22
Timekeepers.....	. 38
Drillmen and helpers.....	3. 15
Miners (for handwork, trimming, etc.).....	. 23
Muckers.....	2. 96
Track and dump men.....	. 74
Mule drivers.....	. 39
Motormen and brakemen.....	. 44
Electricians and blower men.....	. 07
Disabled employees.....	. 19
Timbermen.....	. 22
Miscellaneous.....	. 40
	\$11. 59
Materials:	
Powder, fuse, caps, etc.....	3. 08
Lumber.....	. 29
Oils, candles, etc.....	. 22
Ventilating pipe.....	. 64
Track, including ties.....	. 68
Pressure air pipe.....	. 40
Drill repair parts (including hose).....	. 18
Miscellaneous.....	. 19
	5. 68

	Cost per foot of tunnel.
Repairs:	
Machine-shop expense (including labor and supplies)-----	\$0. 93
Blacksmith-shop expense (including labor and supplies)-----	1. 22
	<u>\$2. 15</u>
Power (all purposes)-----	7. 65
Depreciation:	
Haulage equipment-----	. 09
General equipment-----	1. 00
	<u>1. 09</u>
General expense-----	3. 96
Camp expense-----	1. 21
Corral expense-----	. 25
	<u>5. 42</u>
Total -----	33. 58

"General expense" includes a proportionate charge for the expenses of the Provo office, such as salaries, stationery, telephone, and supplies; also a proportionate charge for the expenses of the Washington, the Chicago, and the supervising engineer's offices. The Provo office covers approximately 68 per cent of this charge, the Washington office 23 per cent, the Chicago office 2 per cent, and the supervising engineer's office 7 per cent.

Detailed cost of driving the west heading for the year 1910, 5,021 feet.

	Cost per foot of tunnel.
Labor:	
Engineering-----	\$0. 61
Superintendence-----	. 60
Shift bosses-----	1. 25
Timekeepers-----	. 22
Drillmen and helpers-----	2. 85
Miners-----	. 28
Muckers-----	2. 93
Track and dump men-----	. 71
Motormen and brakemen-----	1. 49
Electricians and blower men-----	. 13
Disabled employees-----	. 16
Timbermen-----	. 28
Miscellaneous-----	. 07
	<u>\$11. 59</u>
Materials:	
Powder, fuse, caps, etc-----	3. 52
Lumber-----	. 22
Oils, candles, etc-----	. 20
Ventilating pipe-----	. 65
Track, including ties-----	. 74
Pressure air pipe-----	. 28
Drill repair parts (including hose)-----	. 24
Miscellaneous-----	. 07
	<u>5. 92</u>
Repairs:	
Machine-shop expense (including labor and supplies)-----	. 90
Blacksmith-shop expense (including labor and supplies)-----	1. 23
	<u>2. 13</u>
Power (all purposes)-----	<u>5. 70</u>

	Cost per foot of tunnel.
Depreciation:	
Haulage equipment	\$0.20
General equipment	1.00
	<u>\$1.20</u>
General expense	3.32
Camp expense	.63
Corral expense	.08
	<u>4.03</u>
Total	<u>30.56</u>

Detailed cost of driving the west heading, for the year 1911, 3,419 feet.

	Cost per foot of tunnel.
Labor:	
Engineering	\$0.45
Superintendence	.82
Shift bosses	1.65
Timekeepers	.38
Drillmen and helpers	4.07
Miners	.37
Muckers	5.13
Track and dump men	2.00
Motormen and brakemen	1.87
Electricians and blowermen	.08
Disabled employees	.48
Timbermen	1.72
Miscellaneous	.05
	<u>\$19.07</u>
Materials:	
Powder, fuse, caps, etc.	2.61
Lumber	.80
Oils, candles, etc.	.43
Ventilating pipe	.77
Track, including ties	1.52
Pressure air pipe	.36
Drill repair parts (including hose)	.34
Miscellaneous	.25
	<u>7.08</u>
Repairs:	
Machine-shop expense (including labor and supplies)	2.16
Blacksmith-shop expense (including labor and supplies)	1.54
	<u>3.70</u>
Power (all purposes)	5.20
Depreciation:	
Haulage equipment	1.85
General equipment	.50
	<u>2.35</u>
General expense	3.00
Camp expense	1.10
Corral expense	.02
	<u>4.12</u>
Total	<u>41.52</u>

Detailed cost of driving the west heading, January to July, 1912, 2,382 feet.

Labor:	Cost per foot of tunnel.
Engineering	\$0.36
Superintendence	.56
Shift bosses	1.08
Timekeepers	.26
Drillmen and helpers	3.08
Miners	.43
Muckers	4.95
Track and dump men	1.55
Motormen and brakemen	1.33
Electricians and blowermen	.18
Disabled employees	.48
Timbermen	2.59
	\$16.85
Materials:	
Powder, fuse, caps, etc.	2.72
Lumber	2.18
Oils, candles, etc.	.32
Ventilating pipe	.70
Track, including ties	1.51
Pressure air pipe	.30
Drill repair parts (including hose)	.32
Miscellaneous	.39
	8.89
Repairs:	
Machine shop (including labor and supplies)	1.39
Blacksmith shop (including labor and supplies)	1.02
	2.41
Power (all purposes)	3.75
Depreciation:	
Haulage equipment	2.20
General equipment	.50
	2.70
General expense	1.90
Camp expense	.79
	2.69
Total	36.79

Detailed cost of driving the east heading, October, 1911, to July, 1912, 2,682 feet.

Labor:	Cost per foot of tunnel.
Engineering	\$0.49
Superintendence	.77
Shift bosses	1.36
Timekeepers	.31
Drillmen and helpers	3.62
Muckers	4.03
Track and dump men	2.00
Mule drivers	.89
Timbermen	1.80
Electricians and blowermen	.30
Disabled employees	.09
Miscellaneous	.21
	\$15.87

	Cost per foot of tunnel.	
Materials:		
Powder, fuse, caps, etc.....	\$2.67	
Lumber	.98	
Oils, candles, etc.....	.36	
Ventilating pipe	.45	
Track, including ties.....	.56	
Pressure air pipe.....	.12	
Drill repair parts (including hose).....	.38	
Miscellaneous	.21	
		\$5.68
Repairs:		
Machine shop expenses (labor and supplies).....	.62	
Blacksmith shop expenses (labor and supplies).....	.65	
		1.27
Power (all purposes).....		3.21
Depreciation:		
Haulage equipment	.47	
General equipment.....	1.02	
		1.49
General expenses.....	1.86	
Camp expenses.....	1.35	
Corral expenses.....	.95	
		4.16
Pumping (labor and material).....		1.36
Total.....		83.04

THE HISTORY OF TUNNELING.

UNDERGROUND PASSAGEWAYS OF ANCIENT RACES.

The art of excavating underground passageways has been known to mankind for many centuries. The ancient Egyptians and Hindus employed it in the creation of many wonderful subterranean temples and sepulchers in hard rock, and similar monuments are found in the works of the Hebrews, Greeks, Etruscans, Romans, Aztecs, and Peruvians—in fact, of all ancient civilized peoples.

EGYPTIAN TEMPLES AND TOMBS.

It is not surprising that the Egyptians, with their wonderful knowledge of quarrying as well as of many other useful arts, should have been versed in methods of underground rock excavation. Remains of their work, some of which dates back to 1500 B. C., may be found in the grottos of Samoun, the tombs near Thebes and Memphis, the catacombs of Alexandria, and the temples of Ipsamboul. A gigantic tomb has been found at Abydos, which was cut in the solid rock during the XIIth dynasty by Senwosri III; also Rameses II, who is perhaps the best remembered personage of these ancient times, constructed, either because of vanity or the great length of his reign, many rock-cut temples, the grandest of which is probably that of Abu Simbel.

The work was performed with hand tools and the labor necessary to have fashioned monuments of such magnitude and grandeur must have been stupendous. For cutting granite and other hard rock, the workmen used saws of copper which were either fed with emery powder or were set with teeth of that abrasive. A similar method was employed as early as the IVth dynasty for circular holes, which were drilled by a tube having fixed teeth or fed with emery powder. For removing rock in a quarry or in a tunnel, grooves varying in width from 4 to 20 inches were made on four sides of a block, which was then broken out by the swelling action produced by soaking with water a number of wooden wedges driven into these grooves.

HINDU CAVES AND TEMPLES.

The excavations in India probably number at least 1,000, the majority of which are of Buddhist origin. They are usually of two types—chapels and monasteries. The chapels consist of a nave with

a vaulted roof, separated from the side aisles by columns, and containing a small chapel at the inner circular end. The monasteries consist of a hall surrounded by a number of cells for the residence of monks and ascetics.

Most of the Indian excavations are of much later date than those in Egypt. The earliest, the Sudama or Nigope cave, was constructed probably about 260 B. C., the Lomas Rishi was built about 200 B. C., and those of Nassick about 129 B. C. These earlier caves imitated very closely contemporaneous timber-roofed temples, and for this reason the columns all slope inward, copying with great fidelity of detail the rafter supports of the wooden temples. In the Karli caves (about 78 B. C.) this feature is absent; the columns of the nave are quite plumb and the perfection of architecture and ornamentation is unsurpassed by any of the later Hindu rock temples. The galleries and rooms of the caves of Ellora contain a total of nearly 5 miles of subterranean work. Although the builders may possibly have known of gunpowder, it was not used in the construction of these tunnels, which, like all the preceding works, were accomplished laboriously with hand tools and probably by slave labor. The caves of Salsette belong to the sixth century A. D., whereas those at Elephanta were constructed about 800, and the Gwalior temples were excavated still later, during the fifteenth century.

GRECIAN TUNNELS AND MINES.

Modern archaeological investigation indicates that tunneling was possibly known to the Minyæ, an ancient Grecian people dating back beyond 2000 B. C., whose cycle of myths includes, among others, that of the Argonautic expedition. A series of shafts, 16 in all, are to be seen near Lake Kopais, in Bœotia, which are supposed to have been constructed by these peoples for the ventilation of an ancient drainage tunnel. The shafts are 200 to 1,000 feet apart, 6 to 9 feet wide, and have a maximum depth of 100 feet. The tunnel was probably the enlargement of a natural watercourse, such as are commonly found in similar calcareous rocks. Krates, of Chalkis, a mining engineer who lived in the time of Alexander the Great, is credited historically with an attempt to drain this lake by utilizing and enlarging natural watercourses.

Although the exact date of the introduction of mining into Attica, probably from the Orient, is unknown, it seems to have been subsequent to the time of Solon (about 600 B. C.). By 489 it is certain that the silver mines of Laurium were yielding a highly satisfactory return, and at the instigation of Themistocles the net profits from them were applied by the Athenians to the construction of a fleet, so that these mines no doubt contributed largely to the prosperity and

power of Athens. The workings, approximately 2,000 in all, consisted of shafts and galleries in which the rocks were hewn out with hand tools and brought to the surface on the backs of slaves. Air was supplied to the large underground stopes or chambers by ventilating shafts about 6 feet square and 65 to 400 feet deep.

Gold was mined in Macedonia and Thrace at least as early as the fifth century B. C., and Herodotus mentions a tunnel in the island of Samos, built in the sixth century, which was 8 by 8 feet in cross section and nearly a mile long.

AZTEC MINES.

The Aztecs were well acquainted with mining, and they obtained copper from the mountains of Zactollan; the mines of Tasco furnished silver, lead, and tin; and the extensive galleries and other traces of their labor were of great assistance to the early Spanish miners. With no knowledge of iron, although iron ore was very abundant, their best tools were made of an excellent substitute in the form of an alloy of copper and tin. With tools of this bronze they could not only carve the hardest metals, but with the aid of powdered silica they could cut the hardest minerals, such as basalt-porphry, and even amethyst and emerald.

PERUVIAN MINES.

Although the mines of the ancient Peruvians were little more than caverns excavated in the steep sides of the mountains, nevertheless they knew of the art of tunneling, as is shown by the tunnels of their aqueducts and by the extensive tunnel that they built to drain Lake Coxamarco. They, too, had no knowledge of iron, and their tools were made of an alloy of copper and tin, which they probably discovered quite independently of the Aztecs, whom they rivaled also in the cutting of gems.

ROMAN TUNNELS.

The Romans, however, were undoubtedly the greatest tunnel builders of early history. They drove tunnels for passage, drainage, water supply, and mining, not only in Italy but wherever their conquests led them, as is evidenced both by records and by old workings left behind in the countries they dominated. One hardly needs to mention the numerous aqueduct tunnels and sewers of the ancient city of Rome, some of which are in use to-day, attesting the ability of the Romans in this branch of engineering. Remains of their work, many of them remarkably well preserved, have been found in France, Switzerland, Portugal, Spain, Algiers, and even Constantinople.

Their tunnels were of no mean size. A road tunnel near Naples, constructed, according to Strabo, about 36 B. C., was approximately 4,000 feet long, 30 feet high, and 25 feet wide. About 359 B. C. Lake Albanus, which lies about 15 miles southeast of Rome, was tapped for its supply of clear water by a tunnel over 1 mile long, 8 feet high, and 5 feet wide. Possibly the greatest Roman tunnel was driven by the Emperor Claudius to drain the overflow waters from Lake Fucinus, which is situated about 75 miles nearly due east of Rome and has no natural means of outlet. This tunnel, completed in 52 A. D., after 11 years' labor, is over 3 miles long and was designed to be 19 feet high and 9 feet wide; but it appeared to have been even larger than this when, in 1862, it was reopened to obtain valuable land beneath the lake.

These works seem all the more marvelous when one considers the primitive methods available at that time. Explosives were unknown and machinery was not then used in mining. Rock openings were usually made by chipping, by channeling and wedging as in Egypt, or by cutting large grooves around the block to be excavated, using hand tools made of iron, copper, and bronze, although it is quite possible that for certain classes of stone cutting diamonds or some similarly hard minerals were employed in conjunction with primitive tube drills and saws. These methods were often supplemented by fire setting, a method chiefly employed, however, in the large chambers or stopes and not well adapted for driving small tunnels. It consists simply of heating the rock to a very high temperature and quenching suddenly with water (or sometimes with vinegar in calcareous rocks), producing shattering and disintegration because of sudden contraction. Many writers have described the intense and fearful sufferings of men engaged in this work, usually slaves and prisoners of war, who perished by the thousands—a fact, however, of little concern to the ancient builders.

The value of Spain as a storehouse of precious metals, offsetting somewhat the influence of eastern wealth, was well appreciated by Roman leaders, and an armed force for the protection of the mines was maintained there constantly, in many cases at the cost of serious political and financial embarrassment at home. In southern Spain, where the numerous silver and copper mines contained much water, Roman tunnels are very common. They are remarkable for their small size, being usually about 5 feet in height and, where timbered, 16½ to 36 inches in width. One adit, as far as explored, has a length of 1,850 feet and a maximum depth of 183 feet, and another is 2,300 feet long and has a maximum depth of 215 feet.

As nearly as can be ascertained to-day from discoveries in them of various objects of interest, including coins, it is certain that these adits must have been driven very early in the Christian era. Toward

the latter end of the period in which these particular tunnels were used by the Romans attempts were made to work the ore bodies below them by raising water from the lower stopes by means of slave-operated water wheels.

As artificial ventilation by means of blowers was at that time unknown, like most of the Roman tunnels these were ventilated by shafts which were spaced in the tunnel mentioned above at about 25-meter intervals; in order also to minimize the depth to which the shafts were sunk, the courses of the tunnels corresponded very nearly to those of the valleys or gulches above them instead of being straight, as is the usual modern practice. Like the adits, the ventilating shafts were remarkably small. Where timbered the adits were usually about 2 feet 10 inches square in the clear, and where the rock would stand without timbering they were circular and generally did not vary much from 2 feet 4 inches in diameter.

TUNNELING IN EUROPE DURING THE MIDDLE AGES.

With the fall of the Western Empire, tunnel work in Europe practically ceased for many centuries. Some excavations were made, it is true, for tombs and the crypts of monasteries, and underground passages to a secluded exit for escape in time of defeat were a necessary part of the equipment of each castle. Crude attempts at mining also were practiced in Germany. The Teutonic tribes, whose main occupation was warfare and who were barbarous and essentially nomadic at the time of the conquests of Julius Caesar, probably learned from the Romans the value of gold; later they began to search for precious metals and to pursue other peaceful occupations.

During the Middle Ages tunneling was devoted almost exclusively to the needs of war and was seldom employed in constructing aqueducts or other public works. There is, however, a record of a road tunnel begun in 1450 by Anne of Lusignan. It was intended to pierce the Alps at an elevation of nearly 6,000 feet and afford better means of communication between Nice and Genoa, but was never completed. Work was subsequently resumed in 1782 by Victor Amadens III, but was finally abandoned 12 years later after a total of nearly 8,000 feet of tunnel had been constructed.

DEVELOPMENT OF THE USE OF GUNPOWDER IN TUNNELING.

Although gunpowder in Europe, according to the consensus of opinion, was probably invented early in the fourteenth century, and by the end of the sixteenth century was commonly used in military operations for gunnery and for blowing up fortifications, it was not applied directly to mining or tunnel operations during this period. Agricola's "*Bergwerck Buch*," the third edition of "*De Re Metal-*

lica," published by Basel in 1621, a complete English translation of which has been issued, pictures Roman methods and hand work and fire setting as the usual means of mining.

In the year 1613 Martin Weigel is said to have introduced gunpowder in mining. Gatschmann at this time describes the use of wooden plugs for stemming. The plugs were later (about 1685) supplanted by clay. August Bayer ("Das gesegnete Markgrafenthum Meissen," 1732) and Henning Calvör ("Nachrichten über das Berg- und Maschinewesen am Harze, etc.") also confirm the date of 1613 as that of the invention of drilling and blasting, but Honemann and Rössler make it 15 or 20 years later. Whatever may have been the date when blasts were first fired in mines it is certain that blasting had become fairly common by 1650, for powder is mentioned as having been purchased for the Harz mines as early as 1634, drill holes are reported at Dülren which bear the date of 1637, and blasting is known to have been introduced into the Freiberg district in 1643.

The use of gunpowder gave a new impetus to mining and a large number of men became skillful in overcoming the difficulties of underground drifting, so that it is not surprising to note that an increased number of tunnels for other purposes were begun soon after. The chief of these purposes was transportation, and in the eighteenth and early part of the nineteenth centuries many tunnels were driven for canals which, aside from wagon roads, were the only highways at that time. Later the development of steam railroads and the desirability of maintaining level gradients led to the building of a still greater number of tunnels. A brief summary of the features of the more important transportation tunnels constructed abroad and at home follows.

CANAL TUNNELS.

The first modern tunnel constructed for commercial transportation was the Malpas Tunnel on the Languedoc Canal in France. It was 515 feet long, 22 feet wide, and 27 feet high, and was built between 1679 and 1681,^c by Riquet, a French engineer. Although it showed that canals could be constructed through country before thought impassable, no more canal tunnels were driven in France until nearly a hundred years later, the Rive de Gier Tunnel (1,656 feet long) being constructed on the Givors Canal in 1770, and the Torcy Tunnel (3,970 feet long) on the Centre Canal in 1787. The Tronquoy and the Riqueval Tunnels on the St. Quentin Canal were started in 1803, and the Noireu Tunnel (approximately 39,400 feet in length) on the

^c The writers wish to acknowledge their especial indebtedness to Henry S. Drinker, from whose monumental work on tunneling (Tunneling, explosive compounds, and rock drills, 1878, 1025 pp.) this and other valuable information concerning the earlier history of tunnel driving has been obtained.

same canal was begun in 1822. On the Bourgoyne Canal, the St. Aignan Tunnel was started in 1824, so that by the middle of the nineteenth century nearly 20 canal tunnels, with a total length of nearly 93,500 feet, had been constructed in France.

The earliest transportation tunnel in England was the Harecastle, on the Grand Trunk Canal, which was begun in 1766 and opened for traffic in 1777. This tunnel was 8,640 feet in length, 9 feet wide, and 12 feet high. There were originally four other shorter tunnels on this canal. The Harecastle Tunnel was found to be too small to accommodate traffic, and was replaced in 1824 by a parallel tunnel which was 16 feet high and 14 feet wide, 4 feet 9 inches of the width being used for a towpath. The Sapperton Tunnel on the Thames-Medway Canal was started in 1783. It was approximately 12,500 feet long and its construction took six years. The next large canal tunnel in England was the Blisworth (9,250 feet long), on the Grand Junction Canal. It was started in 1798 and required seven years for its completion. In 1856 there were over 45 tunnels on the various English canals, aggregating some 220,000 feet in length.

The first canal tunnel in the United States was the Auburn Tunnel at the Orwisburg landing on the Schuylkill Navigation Canal. The tunnel (which was 450 feet long, 20 feet wide, and 18 feet high) was begun in 1818 and opened for traffic in 1821. The hill it pierced is composed of red shale, and the highest point was only 40 feet above the top of the tunnel. The tunnel was shortened in 1834-37 and again in 1845-46, and was finally made an open cut in 1855-56. The "Summit Level," or Lebanon Tunnel, on the Union Canal, begun in 1824 and finished in 1826, was the second canal tunnel in this country. It was originally 720 feet long, 18 feet wide, and 15 feet high, being driven through argillaceous slate at a total cost of \$30,464. It was followed by the "Conemaugh" and "Grants Hill" Tunnels, on the western division of the Pennsylvania Canal (1827-30); the Pawpaw Tunnel, on the Chesapeake & Ohio Canal (1836); and two tunnels on the Sandy and Beaver Canal, Ohio (1836-38).

RAILWAY TUNNELS.

The first railroad tunnel of which the authors have record was the Terre-noire Tunnel, near St. Etienne, France, on the Roanne-Andrezieux horse railroad. This tunnel, which was begun in 1826, was 4,920 feet long, 9.8 feet wide, and 16.4 feet high. Some 14 other tunnels were built on the road from St. Etienne to Lyons between 1826 and 1833. The first tunnels on a railroad using steam locomotives were those on the Liverpool & Manchester Railway, constructed between 1826 and 1830. It was on this road that the famous trial between the "Rocket," "Novelty," and "Sans Pareil" locomotives

took place in 1829. The following summary of early railroad-tunnel building in Europe is quoted from Drinker's work on tunneling:*

Tunnels, of course, multiplied rapidly in England with the extension of railways, and during the 12 or 15 years following the construction of the Liverpool & Manchester line there were a large number of tunnels built throughout the Kingdom, among them being the famous Kilsby, Box, and Woodhead Tunnels. The first tunnels on a steam railway in France were those built on the St. Germain line in 1837. Subsequently the ones on the Versailles, the Gard, and the Rouen lines raised the total length of tunnels in France in 1845 to 12,833 meters (42,105 feet). The report of the Corps des Ponts et Chaussées on tunnels for 1856 shows at that date a total on French railroads of 126 tunnels, of a total length of 65,106 meters. Among the noted early French tunnels may be cited the Nerthe, Arschwiller, Rilly, La Motte, Lormont, and Alouette. In Belgium the Cumpthieh Tunnel, built in 1835, on the Chemin de l'Etat, seems to have been the earliest. In Germany (Prussia and other States) the earlier lines were so located as to not require much tunnel work; and the Oberau Tunnel (1839), on the Leipzig-Dresden line, in Saxony, was the first. In Austria Rziha gives the Gumpoldskirch Tunnel as the first. A tunnel at Eriebitz (perhaps the same), on the North line, is mentioned in the Ponts et Chaussées report (above cited) as an early Austrian one. In 1856 there were some 50 tunnels in Austria, of a total length of 13,522 meters. In Italy the Naples-Castelamare line, opened in 1840, had several tunnels. In 1856 the total Italian tunnels amounted to 10,181 meters. The Bologna-Pistoja line is especially remarkable for its semisubterranean character. Among the early Swiss tunnels especially to be noted is the Hauenstein, commenced in 1853 and finished in 1858.

The first railway tunnel in the United States was constructed between 1831 and 1833 on the Allegheny Portage Railroad in Pennsylvania. The tunnel (which was driven through slate) was 901 feet long, 25 feet wide by 21 feet high, and was lined throughout with masonry 18 inches thick. It was followed by the Black Rock Tunnel (1835-37) on the Philadelphia & Reading Railroad and the Elizabethtown Tunnel (1835-38) on what is now the Pennsylvania Railroad. After this time railroad-tunnel construction became so general that by 1850 as many as 48 tunnels had been completed on American railways.

MINE TUNNELS.

Among the early European mining tunnels driven with gunpowder and hand drilling mention should be made of the Deep George and the Rothschoenberger Stollen in Saxony, the Joseph II adit at Schemnitz, Hungary, and the Ernest August Stollen, which was later driven under the Deep George. Several tunnels, of which the Tailades Tunnel was the most important, were also driven in connection with the Marseilles Aqueduct during this period.

* Drinker, H. S., *Tunneling, explosive compounds, and rock drills*, 1878, p. 19.

The Deep George Stollen * was driven between 1777 and 1799. The total length of the main tunnel is 34,529 feet. Its various branches aggregate 25,319 feet more, and yet this immense undertaking, driven entirely by hand, was to obtain a drainage depth of only 460 feet. It passed through graywacke for nearly the entire distance.

Work began on the Joseph II mining adit, at Schemnitz, Hungary,^b in 1782, but owing to various interruptions the tunnel was not completed until 1878. The portal is at Wornitz, on the left bank of the River Gran, about 10 miles west of Schemnitz. The tunnel is 10.27 miles long, 9 feet 10 inches high, and 5 feet 3 inches wide, and cost \$4,860,000. It is used entirely for mine drainage, and the annual saving in pumping amounts to more than \$75,000.

The Rothschoberger Stollen * was driven to drain the mines of Freiberg, Saxony; it was begun in 1844 and completed April 12, 1877. The tunnel starts in the Triebisch Valley, at Rothschoberg, about 12 kilometers above Meissen, on the Elbe. Its length on the line planned to Halsbrucke was 42,662 feet, but as completed to a connection with the Hirmelfahrt mine was, including branches, 95,149 feet. The depth below the Anna Stollen was 308 feet. Hand drilling and black powder were used to the end of 1875, when Burleigh drills were introduced. The work was carried on by the State. The tunnel was 9 feet square and was driven from 18 headings, yet 33 years were required for its completion, the average rate of progress in each of the headings being only about 15 feet per month.

The Ernest August Tunnel * was driven below the Deep George Stollen in 1851-1864. The main tunnel is about 34,218 feet long, but the entire length of the adit and its branches is 74,452 feet, all driven in rock similar to that in the George Stollen. The tunnel is 11 feet high and 5½ feet wide, and is driven on a grade of 35.6 feet to the mile. Hand drilling and black powder were used, and with 7-hour shifts, the rate of progress was 50 feet per month; 4-hour shifts increased the rate of progress to 78.7 feet per month, and by crowding the miners to the limit the progress during the last three weeks was 75 feet, or at the rate of 107 feet per month.

Some idea of the importance the early German miners attached to drainage may be gathered from the fact that this colossal enterprise gave them an increased drainage depth of only 315 feet.

The Taillados Tunnel * on the Marseilles Aqueduct was begun in January, 1839, and completed at the close of 1846. It was driven from 14 shafts, and in their construction so much water was en-

* Drinker, H. S., op. cit., p. 351.

^b Wochens, oesterreich, Ing. Achitek. Ver., 1886, p. 284.

* Raymond, R. W., The Rothschoberger Stollen: Trans. Am. Inst. Min. Eng., vol. 6, 1877-78, pp. 542-558.

^d Drinker, H. S., Tunneling, explosive compounds, and rock drills, 1878, p. 351.

^e See Martin, Felix, M. de Mont. Richer et le Canal de Marseille, 1878.

countered that the work of sinking was difficult and at times seemed almost impossible. It was finally necessary to place at one of the shafts a steam engine of 100 horsepower in order to remove the water, which amounted to 3,300 gallons per hour. The cost of sinking the shafts was approximately \$40 per foot, and the tunnel itself cost approximately \$37 per foot, or, including the cost of the shafts, \$48.50 per foot. The Assassin Tunnel on the same project was somewhat less difficult to drive and cost only \$25.50 per foot for 11,400 feet, whereas the Notre Dame Tunnel, which was lined with masonry for its entire length of 11,500 feet, cost \$32.50 per foot.

The first large mining tunnel in the United States was begun as early as 1824. This was the "Hacklebernie" Tunnel near Mauch Chunk, Pa.; it was driven by hand, and black powder was used. When work stopped in 1827 an opening 16 feet wide by 8 feet high had penetrated 790 feet through hard conglomerate. Work was resumed in 1846, and the tunnel was extended to a length of 2,000 feet.

DEVELOPMENT OF THE USE OF ROCK DRILLS AND HIGH EXPLOSIVES IN TUNNELING.

The invention of drilling machines, which occurred almost simultaneously with the discovery of high explosives, gave another great impulse to tunnel driving. The following table gives in chronological order some of the more important events connected with these two wonderful improvements.

A short chronological history of the development of high explosives and rock drills.

- 1847. Sobrero discovered nitroglycerin.
- 1849. J. J. Couch, of Philadelphia, patented on March 29 the first percussion rock drill.
- 1851. J. W. Fowle, of Philadelphia, patented on March 11 the first direct-action percussion drill.
- 1854. Schumann invented his percussion drill at Freiberg.
- 1857. Schumann drills used in Freiberg mines.
- 1857. Sommeiller invented a rock drill for use at Mount Cenis.
- 1861. January 1 Sommeiller improved drills commenced work in the Mount Cenis Tunnel.
- 1863. Nobel first applied nitroglycerin as a blasting agent.
- 1865. Guncotton tried at the Hoosac Tunnel by Thomas Doane, chief engineer.
- 1866. Nitroglycerin tried with great success at the Hoosac Tunnel by T. P. Shaffner.
- 1866. Burleigh drills tried and proved to be a great success at the Hoosac Tunnel.
- 1867. Nobel invented dynamite.
- 1868. Dynamite patented in America by Nobel.

The first extensive utilization of these aids was in the construction of the Mount Ceniz Tunnel in Europe and the Hoosac and Sutro Tunnels in this country. The success attained with them soon led to further activity in tunneling, not only for railroads but in connection with mining, drainage, and water supply as well—an activity culminating in the immense amount of such work undertaken within the last 10 or 15 years.

THE SUTRO TUNNEL.

The idea of draining the mines of Virginia City by a deep tunnel was first broached in the spring of 1860, when Mr. Adolph Sutro began negotiations with the mines, the State, and finally with the Federal Government for contracts, concessions, etc. Actual work was first begun at the portal of the tunnel in Carson Valley, $3\frac{1}{2}$ miles from Dayton, on October 19, 1869. The work was carried on by hand until September, 1872, when diamond drilling was begun and tried rather unsuccessfully. In 1874 Burleigh drills were introduced, operated by compressed air generated in a compressor made by the Société John Cockerill, of Seraing, Belgium. The tunnel was completed July 18, 1878, when the Savage vein was cut 20,000 feet from the portal and 1,922 feet below its outcrop. The tunnel inside of the timbers was 10 feet high by 14 feet wide, divided into two passageways by a central row of posts. The rate of progress varied greatly, ranging from 19 to 417 feet per month, the average monthly rate from start to finish being 192.3 feet.*

THE TEQUIXQUAC TUNNEL.

The Tequixquac Tunnel, which now forms the most important link in the drainage system of the Valley of Mexico, was begun during the reign of the Emperor Maximilian. The work was stopped, however, at the fall of the Empire and was not resumed until 1885; even then the want of funds prevented any material progress until March, 1888.

This tunnel is $6\frac{1}{2}$ miles in length, driven through a mass of sand, mud, and soft calcareous sandstone. It is brick-lined throughout, the section is ovoid, with an extreme width of 13 feet 9 inches and a height of 14 feet, and the tunnel has a gradient of 1 foot in 1,388. The calculated flow of water is 450 feet per second, or 200,000 gallons per minute. At first the headings were driven in the center, but the bottom-heading system was soon adopted of necessity. The greatest completed tunnel advance in any one month was 182 feet, and the

* Report of commissioners, Sutro Tunnel, 1872, 987 pp., and Drinker, H. S., Tunneling explosive compounds, and rock drills, 1878, pp. 337-350.

greatest distance that any single heading was driven in a calendar month was 656 feet.^a

THE SHOSHONE TUNNEL.

The Shoshone Tunnel, 1906-1910, is owned by the Central Colorado Power Co. Its intake portal is on the Grand River 12 miles above Glenwood Springs. The tunnel is 12,453 feet long, 12 feet high, and 16 feet 8 inches wide, and is driven wholly through hard metamorphic granite.

Where timber supports were necessary vertical posts and a three-piece arch were employed, all of which were afterwards completely covered by concrete lining. Driving was carried on from seven cross-cut adits, as well as from both the intake and the discharge ends.

The cost of the tunnel, not including concrete lining, \$927,653, was divided as follows:

Construction costs of Shoshone Tunnel per linear foot of tunnel.^b

Test drifts.....	\$0. 45
Drilling and blasting.....	20. 66
Trenching and grading floor.....	1. 15
Track work.....	1. 76
Mucking and loading.....	17. 28
Hauling.....	2. 88
Dumping and maintenance.....	2. 18
Blasting supplies.....	8. 35
Drill steel.....	2. 91
Sharpening and repairing.....	4. 60
Timbering, temporary and permanent.....	3. 87
Light and wiring.....	1. 57
Ventilating.....	. 59
Pipe, air hose and connections.....	. 85
Power drills.....	2. 94
Holsts and trestles.....	. 96
Pumping.....	. 21
Sundries.....	. 28
Total construction costs.....	74. 49
Overhead costs, including surveying, management, office, etc.....	30. 91
Total cost per linear foot.....	105. 40

TUNNELS IN JAPAN.

Following is a list of some of the more important Japanese mining tunnels:

The Tsude adit, Ashio mine, Shimotsuke district, was driven between 1885 and 1895. It is 11 feet high, 13 feet wide, and 10,000 feet

^a Campbell, A. J., and Abbott, F. W., Tequixquac Tunnel, Valley of Mexico: *Trans. Am. Soc. Civ. Eng.*, vol. 32, pp. 171-194.

^b Data furnished by L. P. Hammond, manager.

long, and is situated on the bank of the Watarase River, at Ashio. The mine contains an aggregate of 540,846 feet of levels.

The Omodani mine, Ono district, has five levels, aggregating 58,380 feet in length, the longest being 12,110 feet, whereas the drainage adit is 10,850 feet long.

The Yoshioka mine, Kawakami district, is opened by eight levels and crosscuts, totaling 134,281 feet, the main adit being 39,198 feet in length.

The Okawamae adit, which was constructed to drain the Kusakura mine, Nugataken, has a length of 10,000 feet.

The Sosuido adit drains the Innai silver mines and is 8 feet high, 10 feet wide, and 7,800 feet long.

SIGNIFICANT DATA FROM RAILROAD TUNNELS.

Although this report deals chiefly with the construction of mining tunnels, much can be learned from the study of tunnels driven for railroads. Under ordinary conditions the rate of progress in a railroad tunnel is limited by the speed at which the advance heading can be driven, and as these headings do not differ materially from mining tunnels the rates of progress attained in them are of great interest to the miner. A railroad-tunnel heading must be driven to line and grade, like a mining tunnel, and although the maintenance of a uniform width and height is desirable, it is not absolutely necessary, railway-tunnel headings having thus a slight advantage over mining tunnels. On the other hand, the multifarious operations carried on between the heading and the portal of a railroad tunnel, even under the best possible organization, often temporarily obstruct transportation; the continuity of the work is sometimes hindered by the shooting of the benches back of the face; and even when all the holes in the benches and headings are blasted together more time is needed to clear out the smoke from so many groups of shots than from a single round in the heading, as in a mining tunnel. On the whole, in similar rock and with equally good equipment and organization there should be little, if any, difference between the speed attained in driving a mining tunnel and a railway-tunnel heading, because, although the conditions for rapid progress are not exactly identical, the opportunities are practically equal.

RATE OF PROGRESS AND COST OF RAILWAY TUNNELS.

The history of the more important railway tunnels of the world also shows forcibly the rapid increase in the rates of driving and the lessening of the cost of construction since the introduction of rock drills and high explosives. The data in the following table are representative.

Progress and cost of some famous railway tunnels.

Tunnel.	Construction period.	Length.	Duration of boring.	Average daily progress in headings.	Cost per linear foot.
		<i>Miles.</i>	<i>Months.</i>	<i>Feet.</i>	
Mount Cenis.....	1857-1870	7.97	157	4.4	\$356.00
Hoosac.....	1858-1874	4.75		a 3.0	396.00
St. Gothard.....	1872-1882	9.26	88	6.2	231.00
Arlberg.....	1890-1893	6.2	40	13.6	182.30
Simplon.....	1898-1906	12.4	78	b 13.69	230.40
Loetschberg.....	1906-1911	9.3	54	c 14.2	211.00

a Average of east and west headings, 1865-1873.

b Allowing only for days on which drilling was carried on, advance was 17.45 feet per day.

c Average for last 30 months, 17.1 feet.

MOUNT CENIS TUNNEL.

The Mount Cenis Tunnel was driven through the northern spur of the Cottic Alps to afford direct connection between the French and Italian railway systems. Work was begun on August 18, 1857, and the French and Italian headings met on December 25, 1870. The length of the tunnel as completed was 42,157 feet and the cost \$15,000,000, or \$356 per linear foot. The greatest depth below the surface was 5,275 feet, where the rock temperature was 85° F. The Sommeiller rock drill, driven by compressed air, was first used in this tunnel January 12, 1861, or five years before the introduction of air drills into the Hoosac Tunnel in the United States. The rate of progress varied greatly with the rock encountered, the total time consumed in driving being 13 years and 1 month, or an average daily progress in each heading of 4.4 feet.^a

HOOSAC TUNNEL.

One of the most important early tunnels driven in the United States was the Hoosac, on the line of the Troy & Greenfield Railway. The project first came under consideration in 1825, but actual work did not begin until 1858. Hand drilling was employed until October 31, 1866, when Burleigh rock drills were first introduced; two months later nitroglycerin was substituted for black powder and the net result of these two most important improvements was greatly to increase the rate of driving. Many disheartening delays and interruptions occurred, due chiefly to failure of the earlier types of drilling machines and to change of engineers and contractors, but in March, 1869, a contract was let to Shanly Bros., of Toronto, who completed the work on December 22, 1874.

The tunnel had a total length of $4\frac{1}{4}$ miles, and most of it was driven through mica schist. The maximum speed attained in a single heading was 184 feet in one month of 26 working days, and the average speed in the east and west headings for the last six months was 4.2 feet per day. The cost was \$10,000,000, or \$398 per linear foot.^b

ST. GOTHARD TUNNEL.

The great undertaking of driving the St. Gothard Tunnel was made possible by a joint treaty between Germany, France, and Italy, and on May 7, 1872, a contract for the tunnel was let to M. Favre,

^a Drinker, H. S., Tunneling, explosive compounds, and rock drills, pp. 354-357; Vernon-Harcourt, L. F., Alpine engineering: Proc. Inst. Civ. Eng., vol. 95, pp. 249-261.

^b Drinker, H. S., op. cit., pp. 315-337.

of Genoa, who gave a bond for \$1,600,000 for the successful completion of the work within a period of eight years. The tunnel is 48,887 feet, or 9.26 miles, in length, and for the most part is through schists. After tests of a number of drills, Ferroux drills were selected for the north side and McKean for the south side. The average rate of progress in the headings was 186 feet per month. In 1880 one of the headings passed through a zone of softened feldspar which, under the weight of the superincumbent rock, squeezed into the tunnel with such force that granite walls and arches 6 feet 7 inches in thickness were required to hold it in place. The maximum rock temperature encountered was 88° F., at a point 5,575 feet below the surface. The headings met February 29, 1880, but the tunnel was not completed until 1882, nearly two years after the time called for in the original contract. The total cost was £2,327,000, or \$231 per linear foot.*

ARLBERG TUNNEL.

The success of the Mount Cenis and St. Gothard Tunnels, coupled with the desire of the Austrian Government to have a railway route to France that would not pass through Germany or Italy, led to the construction of the Arlberg Railway, which runs from Innsbruck, in the Tyrol, to Bludenz, near the Swiss frontier, a distance of 85 miles, piercing the Arlberg Range about 20 miles from Bludenz by a tunnel over 6 miles long. In the selection of the machinery and in planning the work advantage was taken of the experience gained in the Mount Cenis and St. Gothard Tunnels. In consequence the results obtained were as much in advance of those in the St. Gothard as the operations in that tunnel had been an improvement on those employed in the Mount Cenis. The driving of the Arlberg Railway Tunnel began in July, 1880, and the headings met on November 13, 1883, the average rate of progress being nearly 2 miles a year. The greatest temperature of the rock was 64° F., at a point 2,295 feet below the surface. Ferroux percussion drills, operated by compressed air, were employed in the eastern heading and Brandt rotary drills, worked by water pressure, in the western. The Ferroux drills drove 17,355 feet and the Brandt drills 14,880 feet, a difference of 2,475 feet in favor of the former. This variation was due more to the rock in the east and west headings being different than to any difference in the efficiency of the drills themselves, as is shown by the following figures of the average daily advance of the two drills:

* Drinker, H. S., op. cit., pp. 359-370; Vernon-Harcourt, L. F., *Alpine engineering*: Proc. Inst. Civ. Eng., vol. 95, pp. 261-268.

Average daily advance of two types of drills, Arlberg Railway Tunnel.

Year.	Ferroux.	Brandt.
	<i>Feet.</i>	<i>Feet.</i>
1881.....	13.5	9.5
1882.....	17.2	15.1
1883*.....	17.85	17.82

* Ten and one-half months.

The figures show that as the nature of the rock became similar when the faces approached each other the efficiency of the Brandt drill was practically the same as that of the Ferroux. The Brandt drill was much more cheaply operated than the Ferroux and required the use of only 7 miners in the heading, as against 12 with the Ferroux.

The total length of the tunnel was 32,235 feet and its cost was \$5,877,684, or \$182.30 per linear foot.*

SIMPLON TUNNEL.

The Simplon Tunnel consists of two parallel, single-track, railway tunnels, 56 feet from center to center, driven from Brigue, Switzerland, to Iselle, Italy, a distance of 12.4 miles.

Operations began at Brigue November 22, 1898, and at Iselle December 21, 1898. The headings met February 24, 1905, but the tunnel was not completed and ready for use until January 25, 1906. Brandt rotary hydraulic drills were employed in both headings and the average rate of heading advance was 18.69 feet per diem, although when conditions were favorable speeds of 16 feet per diem in the Italian end and 20 to 21 feet in the Swiss end were readily attained. The rock was principally gneiss, with occasional beds of slate, granite, and marble.

In hard rock the cycle of operations was as follows: Bringing up and adjusting drills, 20 minutes; drilling, 1½ to 2½ hours; charging and firing, 15 minutes; mucking, 2 hours.

More serious difficulties were encountered in driving this tunnel than in any other tunnel yet undertaken. Swelling ground was extremely common, and in places the pressure was so great that the roof and sides could be held in place only by steel I beams, with the spaces between rammed with rapid-setting concrete. A part of the tunnel where the pressure was the greatest is said to have cost \$1,620 per linear foot. Many springs were encountered, and at times 17,000 gallons of cold water per minute flowed into the tunnel. Near the center of the tunnel large springs of hot water with a

* Vernon-Harcourt, L. F., *Alpine engineering*: Proc. Inst. Civ. Eng., vol. 95, pp. 268-271; Charton, A. P., *Le chemin de fer et le tunnel de l'Arlberg*: Le Génie Civil, vol. 6, 1885, pp. 3-18.

total flow of 4,330 gallons per minute were encountered. One spring alone (temperature 116° F.) flowed 1,400 gallons per minute. The high temperatures engendered threatened to prevent further advance, but by bringing both cold water and cold air into the headings the temperature was reduced enough to permit work being resumed, although it took six months to drive the last 800 feet. The rapid average rate of progress maintained in the Simplon Tunnel, in spite of the difficulties encountered, was due to superb equipment and an organization so efficient and thorough that 648 men and 29 horses at the Swiss end and 496 men and 16 horses at the Italian end were advantageously employed.

Notwithstanding the care taken in ventilation and the precautions adopted for the health and safety of the workmen, 60 men were killed during the progress of the work. The total cost of the tunnel was £3,200,000, or \$239.40 per linear foot.*

LOETSCHBERG TUNNEL.

The Loetschberg Tunnel was driven through the Bernese Alps in Switzerland, and forms the last link in the railway system connecting the city of Berne with the village of Brigue at the north end of the Simplon Tunnel. The desirability of connecting the Bernese Oberland with the Rhone Valley was discussed as early as 1866, and the present location of the tunnel was first proposed in 1889.

The railway begins at Frutigen in the Bernese Oberland, about 32.5 miles from the north portal; 50.5 per cent of this length is on horizontal curves. There are about 12 short tunnels on the line, aggregating 16,000 feet in length, one of which is a spiral tunnel 5,460 feet long with a 985-foot radius. The main tunnel is 47,678 feet long and was first planned to be run on a tangent, but a serious cave 1.6 miles from the north portal, which killed 25 men and filled 5,900 feet of tunnel, compelled the abandonment of the original line and the adoption of a curved tunnel to pass around the immense, peaty, mud-filled fissure that the heading had tapped.

At the south end Ingersoll-Rand air drills and compressors were used, whereas at the north end Myers drills and compressors were adopted. Compressed-air locomotives running on 30-inch gage tracks were used. In each heading four to six drills were employed, each mounted on a horizontal bar on a carriage, so that mucking out after firing was necessary before drilling could be commenced in the face. For the last 30 months of driving the average rate of progress in the south heading was 15.8 feet per day and in the north end, where the driving was much easier, 18.6 feet per day. On the north side, when

* Saunders, W. L., Tunnel driving in the Alps: *Trans. Am. Inst. Min. Eng.*, vol. 42, pp. 441-446; Fox, Francis, *The Simplon Tunnel*, *Proc. Inst. Civ. Eng.*, vol. 168, pp. 61-83.

the heading was in limestone, it was advanced 5,623 feet in 6 months, or an average rate of 30.8 feet per day.^a

BUSK-IVANHOE TUNNEL.

The Busk-Ivanhoe Tunnel, on the Colorado Midland Railway between Leadville and Glenwood Springs, is 9,394 feet long, and has an altitude of 10,810 feet at Busk and 10,944 feet at Ivanhoe, making it the second highest railway tunnel in the world. It is driven almost the entire distance in metamorphic granite with some softened shear zones that gave considerable trouble in driving and timbering.^b The tunnel cost \$1,250,000, and 30 men were killed in the progress of the work.

SEVERN TUNNEL.

The Severn Tunnel (1873-1887), which is on the line of the Great Western Railway in England and passes under the estuary of the Severn River, has a length of 4.35 miles and traverses a great variety of rocks, including conglomerate, limestone, carboniferous beds, sandstone, marl, and sand. The most serious difficulty encountered in driving was the great volume of water that came, not so much from the estuary above as from a huge spring on the land side. Several ineffectual attempts were made to bulkhead this spring, and before the tunnel could be successfully driven it was necessary to erect an immense pumping plant with a capacity of 45,000 gallons per minute, but the maximum amount pumped for any considerable period did not exceed 20,000 gallons per minute.^c

TOTLEY TUNNEL.

The Totley Tunnel on the Dore & Chinley Railway, between Sheffield and Manchester, England, is 3.53 miles long. Work was begun in 1888, and the tunnel was ready for traffic in September, 1893. It is driven almost entirely through carbonaceous black shale which contained some beds of sandstone and grit. The progress of the work was greatly impeded by heavy intrushes of water, some of which carried vast quantities of sand and silt. For a time the discharge from the Padley heading amounted to 5,000 gallons per minute. At first the water was carried out of the tunnel in 12-inch pipes, but as these proved insufficient and liable to clog with sand, the headings were closed with water-tight bulkheads and center drains carried in

^a Saunders, W. L., Tunnel driving, in the Alps: *Trans. Am. Inst. Min. Eng.*, vol. 42, pp. 446-469; Bonnin, R., *La Nature* (Paris), vol. 37, 1909, pp. 147-157.

^b (Editorial.) Progress of work on the Busk Tunnel, Colorado: *Engineering News*, Aug. 25, 1892, p. 171.

^c Vernon-Harcourt, L. F., The Severn Tunnel: *Proc. Inst. Civ. Eng.*, vol. 121, pp. 305-308.

from the portal. This work took six weeks, during which the pressure behind one of the dams rose to 155 pounds per square inch.*

ASPEN TUNNEL.

The Aspen Tunnel on the Union Pacific Railway, between Cheyenne and Ogden, although only 5,900 feet in length, is interesting on account of the obstacles encountered in driving, the difficulty of holding back the swelling ground, and the fact that mechanical loading of the broken rock was successfully employed in both headings. The tunnel was driven through carbonaceous shale, containing an occasional stratum of yellow sandstone, that dips 20° to 30° east, and the course of the tunnel is a little south of west. The opening is 22 feet 6 inches high and 17 feet wide in the clear, timbered with 12 by 12 inch timbers with vertical posts capped with a 7-segment circular arch. These timber sets were spaced 2 feet apart, 1 foot apart, or closer, as the weight of the ground demanded. In a part of the tunnel walls of solid 12 by 12 inch timbers would not stand the rock pressure, and the timbers were replaced by 12-inch steel I beams, which sometimes buckled sideways before the concrete filling could be rammed in place.

Small air-driven steam shovels with buckets holding $\frac{1}{4}$ cubic yard were employed for loading cars in the headings and effected a great saving in both time and cost.^b

ARTHUR'S PASS TUNNEL.

Arthur's Pass Tunnel, South Island, New Zealand, sometimes known as the Otiro, is on the line of the New Zealand Government Railway which connects Christchurch on the east with Greymouth on the west coast and pierces the crest of the Southern Alps for a distance of 5 $\frac{1}{2}$ miles. Work began in May, 1898, and the contract called for the completion of the work in a period of five years; price \$5,000,000. Tunnel haulage was at first attempted with 16,000-pound benzine locomotives, but they were discarded on account of uncertain action and the annoying fumes, and electric locomotives were substituted.^c

MONT ROYAL TUNNEL.

The Mont Royal Tunnel, which will be 3 $\frac{1}{2}$ miles long, is now being driven through a mountain immediately behind the city of Montreal by the Canadian Northern Railroad Co. Work was begun in July,

* Rickard, Percy, Tunnels on the Dore and Chinley Railway: Proc. Inst. Civ. Eng., vol. 116, pp. 117-138.

^b Hardesty, W. P., The construction of the Aspen (Wyoming) Tunnel on the Union Pacific Railroad: Engineering News, vol. 47, March 6, 1902.

^c Gavin, W. H., Arthur's Pass Tunnel, New Zealand: Engineering News, vol. 67, May 9, 1912, pp. 870-875.

1912. The rock is medium hard to hard limestone, intersected by numerous trap dikes. In each heading three to four drills, mounted on a horizontal bar, are used. One-half of the machines are equipped with a water-feeding device and take air at 100 pounds pressure. The rock cuts readily and as much as 20 feet 6 inches of hole has been drilled in one hour by a 3½-inch machine.*

OTHER TUNNELS.

Data relative to other noted railway tunnels are presented below :

Data relative to other noted railroad tunnels.^b

"Govi" Tunnel, Genoa-Ronco Railway, Italy, 27,093 feet.

Marianopoli Tunnel, on railway from Catani to Palermo, Sicily, 22,453 feet.

Pracchia Tunnel, on railway between Florence and Bologna, 9,000 feet.

Biblo Tunnel, Italy, 13,907 feet.

Nerthe Tunnel, between Marseille and Avignon, 15,153 feet.

Bailly Tunnel, on railway between Paris and Lyons, 13,448 feet.

Kaiser Wilhelm Tunnel, on the Moselle Railway near Kochem, 13,841 feet.

Standridge Tunnel, between London and Birmingham, 15,383 feet.

Graveholts Tunnel, on line of the Bergen Railway, Norway, 3¼ miles.

Kojak Tunnel, on line of Northwestern State Railway, India, 2¼ miles.

Stampede Tunnel, on Northern Pacific Railway, 9,850 feet.

Cascade Tunnel, on the Great Northern Railroad, through the Cascade Mountains in the State of Washington, 13,413 feet.

Tennessee Pass Tunnel, Denver & Rio Grande Railroad, between Leadville and Red Cliff, length 2,572 feet, altitude 10,230 feet.

Tunnel through the crest of the Andes on railway line between Buenos Aires, Argentina, and Santiago, Chile, 15,880 feet long, altitude 10,500 feet.

The summit tunnel on the Peruvian Central Railway, 3,596 feet long, altitude 15,781 feet.

* O'Rourke, D. J., The Mont Royal Tunnel record: Mine and Quarry, July-August, 1913, p. 730.

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- Turbo blowers and compressors. May 31, 1912, p. 578. Describes a 20-stage machine installed at Manchester, England, and discusses the advantages of turbocompressors.
- Unloading device for air compressors. May 24, 1912, p. 542. Describes a device that, when the compressor is not working at full load, permits a part of the air being compressed in the cylinder to flow back to the atmosphere or the intercooler, as the case may be.

- ENGINEERING.** Free-piston internal-combustion air compressor. March 1, 1912, p. 285. Description of a machine recently developed by Signor Matricardi, Palanza, Italy, in which a heavy piston is propelled from one end of a cylinder to the other and during its motion compresses air in front of it.
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- The selection of explosives used in engineering and mining operations. Bureau of Mines Bull. 48, 1912, 50 pp. States the characteristics of different classes of explosives and sets forth the results of tests showing the suitability of explosives for different kinds of blasting. The bulletin is written for the information of all persons interested in the use of explosives for blasting rock.
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- Hydraulic cartridge for mining. April 20, 1912, p. 364. Describes a cartridge that expands by hydraulic pressure and is useful for breaking rock when it is essential that no shocks be imparted to the surroundings.
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- SNELLING, W. O., and HALL, CLARENCE.** The effect of stemming on the efficiency of explosives. *Bureau of Mines Technical Paper* 17, 1912, 20 pp. The gain in efficiency by the use of stemming was demonstrated by firing small charges of explosives in bore holes in lead blocks. The pamphlet is of interest to all persons who use explosives for blasting coal or rock.

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Bulletin 58

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BUREAU OF MINES

JOSEPH A. HOLMES, DIRECTOR

FUEL-BRIQUETTING INVESTIGATIONS

JULY, 1904, TO JULY, 1912

BY

C. L. WRIGHT



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FUEL-BRIQUETTING INVESTIGATIONS, JULY, 1904, TO JULY, 1912.

By C. L. WRIGHT.

INTRODUCTION.

In 1904 the Government began a series of fuel-testing investigations at its fuel-testing plant at St. Louis, Mo. These investigations, which were placed under the supervision of the United States Geological Survey, included chemical analyses, steaming tests, gas-producer tests, briquetting tests, coking tests, and washing tests of samples collected from the various coal fields of the country. The specific purpose of the investigations was to analyze and test the coals and lignites in the United States, in order to determine their fuel values and the most efficient methods of utilizing them for different purposes.

Part of the fuel-testing plant, including the briquetting equipment, was moved to Norfolk, Va., in 1907, and the fuel-testing investigations were made one of the divisions of the technologic branch of the Geological Survey. In 1908, after the close of the Jamestown Exposition, most of the fuel-testing equipment was removed to its present location at the Pittsburgh experiment station, Fortieth and Butler Streets, Pittsburgh, Pa. In 1910, an act of Congress, effective July 1, established the Bureau of Mines in the Department of the Interior and authorized the transfer to this bureau of the fuel-testing investigations that had been carried on by the technologic branch.

Bulletins presenting the results of various inquiries and investigations relating to briquetting have been published by the Geological Survey and the Bureau of Mines.^a As the supply of some of these bulletins is exhausted, the Bureau of Mines has decided, instead of reprinting them, to publish the more important facts they contain

^a Parker, E. W., Holmes, J. A., and Campbell, M. R., committee in charge, Preliminary report of the operations of the coal-testing plant at the Louisiana Purchase Exposition, St. Louis, 1904: U. S. Geol. Survey Bull. 261, 1906, 172 pp.; Holmes, J. A., Preliminary report on the operations of the fuel-testing plant of the United States Geological Survey at St. Louis, Mo., 1906: U. S. Geol. Survey Bull. 290, 1906, 240 pp.; Parker, E. W., Holmes, J. A., and Campbell, M. R., committee in charge, Report on the operations of the coal-testing plant of the United States Geological Survey at the Louisiana Purchase Exposition, St. Louis, Mo., 1904: U. S. Geol. Survey Prof. Paper 48, 1906 (in three parts), 1492 pp., 13 pls.; Holmes, J. A., Report of the United States fuel-testing plant at St. Louis, Mo., January 1, 1906, to June 30, 1907: U. S. Geol. Survey Bull. 332, 1908, 299 pp.; Mills, J. E., Binder for coal briquets (investigations made at the fuel-testing plant, St. Louis, Mo.): U. S. Geol. Survey Bull. 343, 1908, 56 pp. (reprinted as Bull. 24, Bureau of Mines); Wright, C. L., Briquetting tests at the United States fuel-testing plant, Norfolk, Va., 1907-8: U. S. Geol. Survey Bull. 335, 1909, 41 pp., 9 pls. (reprinted as Bull. 30, Bureau of Mines); and Wright, C. L., Briquetting tests of lignite at Pittsburgh, Pa., 1908-9, with a chapter on sulphite-pitch binder: Bull. 14, Bureau of Mines, 1911, 64 pp., 11 pls., 4 figs.

and the results of all other briquetting investigations undertaken by the Government from July 1, 1904, to July 1, 1912, in one volume. Material from other bulletins containing reports of steaming tests of briquets is included, due credit in each case being given.

GENERAL EQUIPMENT OF GOVERNMENT FUEL-TESTING STATIONS.

The equipment for carrying on briquetting tests at the St. Louis fuel-testing plant consisted of three briquetting plants—one exhibited by William Johnson & Sons, of Leeds, England; one by the American Compressed Fuel Co., of Chicago, Ill.; and one by the Renfrow Briquet Machine Co., of St. Louis, Mo. These presses or plants are referred to in the text as the "English machine," the "American machine," and the "Renfrow No. 1 machine." A building was provided for the English plant and another for the American plant. Steam and electric power were furnished by the power plant of the station. A direct-heat cylindrical drier contributed by the C. O. Bartlett & Snow Co., of Cleveland, Ohio, was used for drying the washed coals and lignites when the moisture content was too high to permit successful briquetting. A Williams crusher contributed by the Williams Patent Crusher & Pulverizing Co., of St. Louis, Mo., and a Stedman pin disintegrator made by the Stedman Foundry & Machine Works, of Aurora, Ind., were used for reducing the fuel and binders to sizes suitable for briquetting.

The equipment used for briquetting experiments at the Norfolk plant consisted of the English plant used at St. Louis, a new Renfrow machine designated in this report as the "Renfrow No. 2," and a Schlickeysen "auger" machine for manufacturing "machined peat."

At the Pittsburgh plant the English machine was placed on a foundation, but auxiliary equipment for it has not yet been installed, and most of the briquetting experiments at this station have been made with the German lignite-briquetting plant, complete with press, drier, and cooler. A small hydraulic caking press made by the Watson-Stillman Co. of New York, and operated by hand, was also installed at this station and used for making preliminary briquetting tests of fuels and comparative tests of binders.

PERSONNEL OF BRIQUETTING STAFF.

The various fuel investigations at St. Louis in 1904 were under the general supervision of E. W. Parker, statistician of the Geological Survey; J. A. Holmes, then State geologist of North Carolina; and M. R. Campbell, geologist of the Geological Survey. This committee, with the addition of C. W. Hayes, then chief geologist of the Geological Survey, also had charge of the work in 1905. Since early in 1906 the work has been under the general direction of J. A. Holmes, now director of the Bureau of Mines.

The briquetting tests at St. Louis were originally directed by J. H. Pratt, of the University of North Carolina, assisted by A. A. Steel. Later the tests were placed in charge of C. T. Malcolmson, who was assisted by J. E. Mills, W. J. Chapman, Robert Strasser, G. E. Ryder, Ralph Galt, and C. L. Wright. The tests at Pittsburgh were directed by C. L. Wright, assisted by H. L. Gardener and Otto Lehman. Acknowledgement is gratefully made of valuable advice and suggestions from H. M. Wilson, A. W. Belden, O. P. Hood, and J. C. Roberts.

The chemical analyses were made at the laboratory of the fuel-testing division under the direction of F. M. Stanton and, later, A. C. Fieldner.

COMMERCIAL ASPECTS OF BRIQUETTING IN THE UNITED STATES.

There are some American engineers who doubt whether fuel briquetting can be successful in the United States for many years to come. This skepticism probably is caused by the numerous unsuccessful attempts to briquet fuels on a commercial scale.

CAUSES OF FAILURE OF BRIQUETTING PLANTS.

The failures mentioned were in a large measure due to the following causes:

1. Many of the "plants" represented promoters' schemes, no attempt being made to build the plants.
2. Attempts were made to develop a new binding material or a new press, without proper appreciation of the principles of briquetting.
3. Plants were poorly situated for marketing the briquets.
4. The briquets produced were inferior. Briquets containing an excessive amount of pitch binder have been placed on the market, with the result that after one trial householders were disgusted with the soot and odor produced. Results of this kind are especially harmful to the development of the industry, as it is very hard to overcome a prejudice against briquets when once it is created.
5. Poor salesmanship has been responsible for other failures of briquetting ventures. As briquets are a comparatively new and untried form of fuel careful salesmanship to introduce them is required. As many retail coal dealers are afraid that the new fuel will supplant the usual fuels they do not encourage the introduction of briquets; some even tell their customers that briquets are made of dirt and have practically no heating value. Therefore, to build up a successful business in fuel briquetting requires much advertising and demonstration, not to mention capital and time.
6. Uncertainty in the supply of raw fuel or binder has caused the failure of many briquetting plants.

7. Lack of technical supervision has in some instances been the direct cause of failure. Certain features of briquetting, notably suitable binder, proper temperature, and the influence of moisture, are all problems that require technical investigation and chemical control. The making of satisfactory briquets involves considerably more than the possession of suitable machinery. Slight variations in the quality of binder used may make all the difference between success and failure, as the binder is the largest item in the briquetting cost.

CONDITIONS FAVORING DEVELOPMENT OF THE INDUSTRY.

The condition that more than any other has prevented the development of this industry in the United States is the low price of bituminous coal and especially the small difference between the price of the lump coal and that of the slack, or fine coal. Much of the lump coal is finally crushed before being manufactured into coke, and the slack coal has even the advantage in not having to be crushed. Where these coals are sold for general power purposes much of the fine coal can be shipped and burned with the lump coal, for the reason that it fuses as soon as it begins to burn, and therefore neither clogs the draft nor escapes with the ashes, as is the case with noncoking coals.

In States where noncoking coals are produced the demand for the slack coal is not so great, and in some places it can be bought for less than 50 cents per ton. At a number of mines large quantities are thrown on the dump as waste, and frequently the slack is burned there in order to get rid of it.

A large variety of special equipment for burning fine coal has been developed in the United States. The results of many tests seem to indicate that with the noncoking coals the use of the finely powdered coal, even in the modern furnace, is a disadvantage rather than a gain. The fine material sifts through on to the grates, prevents a full draft, and may be ultimately lost with the ashes. This material should therefore be separated from the coarser coals and made into briquets if possible.

With anthracite and semianthracite coals the difference between the price of the lump coal and that of the slack is often more than sufficient to cover the cost of manufacturing briquets, and in such cases there can be no question as to the possibility of establishing a briquetting industry as soon as the proper binding material and briquetting equipment can be provided.

There are still other cases in which the difference between the price of the lump coal and of the slack is about sufficient to cover the cost of briquet manufacture. The fact that the briquets made from this material present certain advantages over the lump coal make them command a price enough higher to provide a margin of profit.

The high cost of pitch binder is one of the barriers in the way of the development of this industry. One of the purposes of this investigation has been to discover, if possible, some cheaper binding material. The cost of manufacturing briquets in France, Germany, Belgium, and England, including all necessary items except the cost of the coal and binding material, ranges from 25 to 50 cents per ton. Where pitch is used as a binder, its cost for a ton of briquets varies from 50 to 80 cents. How far the cost of briquetting may be reduced by the use of cheaper binders remains to be seen.

PRESENT OUTLOOK.

The most favorable outlook for the development of this industry in the United States is in the direction of the use of briquets in locomotives and in domestic furnaces and stoves. It can hardly be expected that at the present prices of slack coal briquets can be manufactured for successful use in the ordinary power-plant furnaces.

The fuel-briquetting industry has passed the pioneer stage and is making a steady growth each year. The engineering difficulties have been solved to the extent that at least one engineering firm is willing to build a briquetting plant and guarantee it to produce a specified tonnage of briquets of a certain quality, the plant's efficiency to be established before payment is made.

There were in operation in the spring of 1912 18 fuel-briquetting plants, all of which were visited by the writer; about 6 other plants may be operating successfully, although they were not inspected by the writer in 1912.

ADVANTAGES OF BRIQUETTED OVER NATURAL FUEL.

Briquets when properly made with a suitable binder possess the following advantages over raw fuel:

1. The even size of the briquets permits a more regular and thorough combustion in the firebox or furnace.

2. They produce much less smoke, and, in many cases, practically no smoke. This feature of briquets is more noticeable with the smaller sizes than with the large rectangular blocks. On account of the number of flat surfaces the latter tend to pack together, thus preventing free access of air. If an excessive amount of tarry pitch is used some smoke will be given off.

3. Good briquets retain their shape in the fire, and do not cake sufficiently to cut off the supply of air to the upper surface of the fire.

4. Briquets usually burn to a fine ash without clinkering. In the briquetting process the mixing and grinding distributes the ash or foreign material, which forms lumps and layers in the raw coal. When raw coal is burned many of these lumps and layers, being too large to pass the grate, are fused into clinkers.

5. A briquet fire requires much less care than one of raw fuel.

6. The evaporative power of briquets is greater than that of coal in its natural condition. This advantage has been found to exist at all rates of evaporation.^a

8. The weather-resisting qualities of many coals, especially lignites, are greatly improved by briquetting.

9. Steam can be more quickly and easily raised with briquet fuel than with run-of-mine coal.^b

10. Higher rates of combustion are possible with briquets than with run-of-mine coal.^c

11. The loss from breakage during transportation of good briquets is less than with run-of-mine coal. This loss should not exceed 5 per cent.

12. The possibility of spontaneous combustion is eliminated if the fuel is stored in the briquetted form, and for this reason fuel in the form of briquets is favored by European countries.

13. The large block-shaped briquets may be piled in regular rows, and occupy less space per ton than the run-of-mine coal. However, the egg-shaped or cylindrical briquets occupy more space than the run-of-mine fuel.

14. Briquets have a higher heating value than the raw fuel from which they are made, by reason of the higher heating values of the binders and the loss of water, especially in the case of lignite, during briquetting.

THE BRIQUETTING OF AMERICAN LIGNITES.

INTRODUCTORY STATEMENT.

The industrial supremacy of the United States is dependent on its fuel resources. A small part of the total fuel supply of the United States has been used, but increase in the cost of mining will lead to a better utilization of the waste products from the mining industry.

Of the five general classes of fuel, namely, anthracite, bituminous coal, subbituminous coal (also called black lignite), lignite (or brown coal), and peat, anthracite and bituminous coals are in most general use to-day in the United States, the subbituminous being used in the West to an increasing extent. Lignite and peat are at present utilized only as local fuels and in a few regions.

In the western and southern parts of the United States there are large deposits of lignite, which, as it comes from the mine, is not suitable for general use as fuel. The high proportion of water or

^a Goss, W. F. M., Comparative tests of run-of-mine and briquetted coal on locomotives: Bull. 37, Bureau of Mines, 58 pp., 4 pls., 35 figs. (reprint of U. S. Geol. Survey Bull. 363).

^b Ray, W. T., and Kreislinger, Henry, Tests of run-of-mine and briquetted coal in a locomotive boiler: Bull. 34, Bureau of Mines, 33 pp., 9 figs. (reprint of U. S. Geol. Survey Bull. 412.)

^c Ray, W. T., and Kreislinger, Henry, Comparative tests of run-of-mine and briquetted coal on the torpedo boat *Biddle*: Bull. 33, Bureau of Mines, 50 pp., 10 figs. (reprint of U. S. Geol. Survey Bull. 23).

moisture (in some cases as high as 42 per cent) decreases the heat value of the raw lignite; the evaporation of the moisture causes lignite to crumble or slack when exposed to the air for a few days, and to deteriorate greatly during storage or long transportation. Lignite stored in large piles is much more liable to spontaneous combustion than bituminous coal. Lignite is noncoking. As it falls to pieces and readily sifts through the grate, special furnaces are required to burn the raw lignite properly.

Because of its inferiority, lignite has not been used extensively in this country. It has been used in Texas, North Dakota, and other States to some extent as a local household fuel, and is rapidly gaining favor as a gas-producer fuel in the State of Texas. In many localities where lignite is found in abundance, good bituminous coal commands a high price, and for this reason the briquetting of this lignite as a means of improving its heating value and making it available for a household fuel has attracted attention.

German lignite contains a higher percentage of water than American lignite and most of it is so soft that it can be cut with a spade. Many lignite beds in Germany are filled with well-preserved logs and pieces of wood.

EXTENT OF LIGNITE RESOURCES IN UNITED STATES.

The United States Geological Survey has estimated the supplies of coal and lignite in the United States as follows:^a

Tonnage by grades of coal and accessibility.

[Short tons.]

Kind of coal.	Area, in square miles.	Original supply.		Easily accessible coal still available.
		Amount easily accessible.	Amount accessible with difficulty.	
Anthracite and bituminous	250, 531	1, 257, 766, 000, 000*	505, 730, 000, 000	1, 247, 672, 000, 000
Subbituminous	97, 636	356, 707, 000, 000	233, 450, 000, 000	356, 594, 000, 000
Lignite	148, 609	389, 545, 000, 000	354, 045, 000, 000	389, 534, 000, 000
Total	496, 776	2, 004, 018, 000, 000	1, 153, 225, 000, 000	1, 993, 800, 000, 000

A recent estimate^b made by the United States Geological Survey indicates that the lignite resources are greater than given in the earlier estimate. The later estimate follows.

^a Mineral Resources, U. S., 1907.

^b In letter to author.

FUEL-BRIQUETTING INVESTIGATIONS.

Lignite deposits in the United States, short tons.

State.	Area, in square miles.		Estimated original lignite supply in both easily accessible and accessible with difficulty.
	Containing workable beds.	May contain workable beds.	
Alabama.....	(a)	6,000	(a)
Arkansas.....	100	5,900	90,000,000
Louisiana.....	8,800	(a)	(a)
Mississippi.....	7,500	(a)	(a)
Montana.....	33,130		385,474,700,000
North Dakota.....	29,630	6,350	697,929,400,000
South Dakota.....	2,160	820	1,020,300,000
Tennessee.....	1,000	(a)	(a)
Texas.....	2,000	53,000	23,000,000,000
California.....	(a)	(a)	(a)
	156,390		1,087,514,400,000

a Data insufficient on which to base an estimate.

There are many deposits of lignite in the Western and Southern States, the most important deposits being found in Texas, North Dakota, Montana, and California. These deposits vary in thickness from a few inches up to 80 feet.

PRODUCTION AND VALUE OF LIGNITE IN UNITED STATES.

The following table gives the production and value of lignite mined in the United States in 1910:^a

Production and value of lignite in the United States in 1910.

State.	Production.	Average price per ton at mine.
	Tons.	
California.....	11,164	\$1.64
North Dakota.....	595,139	1.49
Oregon.....	67,533	3.48
Texas.....	881,232	.87

BRIQUETTING OF LIGNITE IN THE PAST.

Germany and other European countries have been briquetting lignite successfully for 30 years. The main causes why this industry has not been developed in this country are: The low price of high-class American fuels compared with the price of similar fuels in foreign countries, and the small demand for fuel in the western part of the United States where the larger deposits of lignite are found.

A series of investigations was made with the German machine by the Bureau of Mines without the use of any artificial binder. Those lignites that were not made into good briquets in this series of experi-

^a Compiled from data in "Production of coal in 1910," from Mineral Resources U. S. for 1910, by E. W. Parker, U. S. Geol. Survey.

ments may possibly produce excellent briquets in a machine adapted to working with artificial binders. Preliminary tests with a hand press indicate excellent possibilities in the use of binders in making lignite briquets.

GERMAN BRIQUETTING PRESS.

The German press, mentioned above, is typical of those used in Germany for briquetting lignite without binders. It is of the open-mold type, and pressure is developed by forcing the lignite through a gradually decreasing opening or tube 3 feet long. This press gives a working pressure of 14,000 to 28,000 pounds per square inch. The capacity of the press is about 3.8 tons per hour, but varies with different lignites according to the ease with which they can be briquetted. The speed varies from 60 to 120 revolutions per minute, the latter figure being the rated speed of the machine when working on German lignites.

RESULTS OF LIGNITE-BRIQUETTING TESTS.

All lignite was shipped from the mines to the plant in box cars to eliminate weathering while in transit. The lignite was stored in covered bins and for each test 3 or 4 tons was prepared for pressing by being crushed, dried, and cooled.

As a result of the tests conducted by the Bureau of Mines, using the German method without binder, it was found that briquets could be made of lignite from the following localities:

Texas.—Lytle, Medina County. Satisfactory briquets made, but lignite difficult to briquet.

North Dakota.—Scranton, Bowman County. Satisfactory briquets made. Lehigh, Stark County. Difficult to briquet, but some fair briquets made.

California.—Ione, Amador County. Excellent briquets easily made.

Lignites from the following localities were not briquetted on the German press:

Texas.—Rockdale; sample insufficient. Calvert; no satisfactory briquets made.

North Dakota.—Vanderwalker; no satisfactory briquets made.

After these tests, tests were made with the hydraulic hand press, various binders being used. The results indicate that certain Texas and North Dakota lignites can be made into satisfactory briquets with the percentage of binder given in the following table as the minimum required.

Minimum percentage of certain kinds of binder required with certain lignites.

Sample from—	Binder.	
	Amount.	Kind.
Vanderwalker, N. Dak.....	<i>Per cent.</i> 8	Water-gas pitch.
Do.....	8	Flour. ^a
Do.....	8	Cell pitch. ^a
Calvert, Tex.....	6	Water-gas pitch.
Do.....	3	Flour. ^a
Do.....	8	Cell pitch. ^a

^a Briquets were not waterproof.

INCREASE OF HEATING VALUE.

The following table shows how the heating value of lignite is increased by the removal of moisture during briquetting:

Improvement of heating value of lignite by briquetting.

Source of sample.	Field designation.	Moisture.			Heating value-per pound.		
		In raw lignite.	In briquets.	Re-moved.	Raw lignite.	Briquets.	Increase.
		<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>B. t. u.</i>	<i>B. t. u.</i>	<i>Per cent.</i>
Texas.....	Pittsburgh No. 11.	33.0	9.0	24.0	6,840	9,336	36.5
North Dakota.....	Pittsburgh No. 8.	40.0	12.0	28.0	6,241	9,354	50.0
Do.....	Pittsburgh No. 13.	42.0	10.0	32.0	6,079	9,355	54.0
California.....	Pittsburgh No. 14.	40.0	10.0	30.0	6,080	9,264	52.4

Excessive moisture in fuel not only causes a waste of useful heat during combustion, but also is a source of expense to the consumer, who pays freight charges on useless water. In the case of one of the North Dakota lignites the removal of 32 per cent of moisture during briquetting reduces decidedly the cost of supplying a consumer with a given number of heat units. If the briquets possessed no other advantage over raw lignite than their higher heating value, they would be worth 50 per cent more than the raw fuel. The table shows that the average heating value of the different lots of briquets was about 9,300 British thermal units in each case, although the heating values of the raw fuel varied considerably.

RESISTANCE TO WEATHERING.

The results of the weathering tests indicated that the lignite briquets resist weathering much better than the raw fuel. Some briquet samples after exposure to autumn and winter weather for four months were in nearly as good condition as at first. Samples stored under cover did not show any signs of deterioration after six months, although the raw lignite stored under the same conditions disintegrated in two months.

RESISTANCE TO HANDLING.

The briquets bear handling better than the raw lignite. Their superiority is, of course, more striking after the lignite and the briquets made from it have been stored for several weeks.

EFFICIENCY AS FUEL.

Preliminary results of producer-gas and steaming tests indicate that as gas-producer fuel the briquets were 43 per cent more efficient, and as boiler fuel were 14.5 per cent more efficient, than the raw lignite.

WORK OF NORTH DAKOTA EXPERIMENT STATION.

North Dakota recently passed laws requiring all its public schools to use lignite for heating wherever possible. By act of legislature an experimental plant has been established at Hebron as a substation of the mining-engineering department of the University of North Dakota, and the gas-making and briquetting qualities of the lignites of the State are being investigated by Prof. E. J. Babcock. At present there is no commercial briquetting plant in North Dakota, although there have been numerous promotion schemes projected from time to time within the State. Babcock^a states that none of the various processes that have been hitherto suggested for the treatment of North Dakota lignite has been adapted to the needs of a successful briquetting industry. As a result of his research he believes that North Dakota lignite can not be made of great commercial value except in the immediate vicinity of the mines, unless it can be raised to a higher fuel value. To accomplish this purpose he has developed a method of removing and saving the gas and other by-products and briquetting the residue with a coal-tar pitch and a starchy material as a binder. This residue has a high fuel value, but does not coke, and would therefore be worthless unless briquetted. When briquetted it becomes nearly equal in heating value to the average anthracite. The briquet is strong, stands shipping well, retains its form while burning, and is of such high heating value that the product can be transported to a considerable distance and marketed at a profit.

^a In correspondence with the author.

COST OF BRIQUETTING LIGNITE.

The cost of briquetting lignite should be considered under two heads: (1) Cost of briquetting without binder by the German method; and (2) cost of briquetting with binder on low-pressure machines.

COST BY THE GERMAN METHOD.

In Germany $1\frac{1}{2}$ to 2 tons of raw lignite is required to make 1 ton of briquets. Franke^a gives figures on the cost of briquetting lignite in a German plant having two presses and a capacity of 120 long tons per 24 hours or 3,000 tons per month. The following table was taken from Franke's work, the German weights and costs having been converted into American units for easier comparison:

Cost of briquets without binder in Germany.

Item.	Monthly.	Per ton of briquets.
1. Interest and depreciation:		
(a) Interest, 5 per cent on \$111,000	\$462.00	} \$0.400
(b) Depreciation—		
10 per cent on \$66,000, machinery and equipment.....	550.00	
5 per cent on \$45,000, building.....	188.00	
2. Raw lignite:		
(a) Lignite for 1 ton briquets, 68 bushels, at 1.05 cents per bushel.....		.712
(b) Lignite for boiler fuel, for 1 ton briquets, 34 bushels, at 1.05 cents per bushel.....		.356
Total, 102 bushels, at 1.05 cents, or monthly, $3,000 \times 102 \times 1.05$ cents.....	3,215.00	
3. Labor for 24 hours (2 shifts):		
2 engineers, at 83.3 cents.....	\$1.66	
2 firemen, at 83.3 cents.....	1.66	
1 ash remover, at 71.4 cents.....	.71	
3 coal passers, at 83.3 cents.....	2.50	
8 drier fillers, at 83.3 cents.....	6.66	
2 pressmen, at 83.3 cents.....	1.66	
1 machinist, at 83.3 cents.....	.83	
12 boys for loading and piling, at 53.3 cents.....	6.40	
31 men, total pay per day.....	22.08 $\times 25$	
2 foremen, 1 weighman, at \$47.60 per month each.....	552.00	} .231
4. Oil, waste, and lighting material.....	142.50	
5. Repairs, maintenance, etc.....	176.00	
	125.00	.042
Total.....		1.800

The cost of briquetting without binder by the German method will of course vary to some extent with the location of the plant. The fact that lignite is high in moisture makes it imperative that the plant should be at the mine. In the case of a plant for briquetting bituminous or anthracite coals, the plant may be erected to better advantage away from the mines and near the point of consumption of the briquetted fuel.

COST IN THE UNITED STATES.

Considering now the equivalent briquetting cost in the United States, using an imported German machine and the equipment necessary, all imported, there will have to be added to the German cost of \$66,000 for a plant a duty of 45 per cent, or \$29,700.

^a Franke, G., *Handbuch der Briquetbereitung*, vol. 1, 1910, p. 591.

The cost of a two-press briquetting plant, equipped with German presses and driers, but in which American auxiliary equipment is used, and all housed in a building of steel and concrete walls, should not exceed \$100,000. Such a plant would have a capacity of about 100 tons of briquets per day of 20 hours, or 2,500 tons per month of 25 working days. With this equipment the cost of production in America would be as follows:

Cost of briquets without binder in United States.

Item.	Monthly.	Per ton of briquets.
1. Interest and depreciation:		
(a) Interest, 5 per cent on \$100,000.....	\$417.00	} \$0.458
(b) Depreciation—		
10 per cent on \$75,000, machinery and equipment.....	625.00	
5 per cent on \$25,000, buildings.....	104.00	
2. Raw lignite:		
(a) For briquetting 1.5 tons for 1 ton briquets, at \$1.50 per ton.....		2.25
(b) Lignite for boiler fuel, 0.5 ton for each ton briquets, at \$1.50 per ton.....		.75
Total, 2 tons, at \$1.50, or monthly, 2,500×2×\$1.50.....	7,500.00	
3. Labor for 20 hours (2 shifts):		
2 engineer-pressmen, at \$3.....	\$6.00	
2 firemen, at \$1.75.....	3.50	
2 coal unloaders, at \$1.75.....	3.50	
2 briquet loaders, at \$1.75.....	3.50	
2 roustabouts, at \$1.75.....	3.50	
1 machinist, at \$3.....	3.00	
11 men, total pay per day.....	23.00×25	
1 clerk or weighman, at \$50 per month.....	50.00	.250
4. Oil, waste, and light.....	100.00	.04
5. Repairs, maintenance.....	125.00	.05
Total.....		3.878

The cost of briquets made in the United States, with raw lignite at \$1.50 per ton at the mines, would be about \$3.88 per ton. As 2 tons of raw lignite is used, the cost of the raw fuel is 77 per cent of the total cost.

Using the values given on page 8, the price of Texas lignite at the mine being averaged at 87 cents per ton, the cost of briquets would be:

2 tons lignite, at 87 cents.....	\$1.74
Briquetting costs.....	.88
Total, per ton.....	2.62

If North Dakota lignite were used, the cost of briquets would be:

2 tons lignite, at \$1.49.....	\$2.98
Briquetting costs.....	.88
Total, per ton.....	3.86

If California lignite were used, the cost of briquets would be:

2 tons lignite, at \$1.64.....	\$3.28
Briquetting costs.....	.88
Total, per ton.....	4.16

As the above figures are based on the value of the lignite at the mine, the cost of briquets would be somewhat less if the briquetting were done by the operator of the mines, as the raw material used might include slacked lignite, which would generally be considered valueless.

From a comparison of the cost of briquets made in Germany with the cost of those made in the United States, it will be noticed that the briquetting cost in Germany, exclusive of the raw fuel used, is approximately 73 cents per ton of briquets, and in the United States it is 88 cents per ton. This increased cost in the United States is due to the higher cost of labor and the shorter working hours (10 instead of 12). The cost would be higher still in the United States if it were not for the greater efficiency per man and the use of mechanical handling of materials. It will be noticed also that the cost of the raw lignite needed to produce 1 ton of briquets in Germany is \$1.07, whereas in the United States the equivalent quantity costs \$3.

MARKET PRICE OF BRIQUETS IN EUROPE.

A consular report of 1907 gave the following figures of the market price of briquets in Europe:

Market price of briquets in Europe.

Location.	Kind of briquet.	Size or shape (inches).	Weight.	Price.	
				Wholesale.	Retail.
Dresden, Saxony...	Brown coal....			\$1.19-\$1.178 per 1,000.	\$0.15-\$0.20 per 100.
	Bituminous....	5 by 3 by 5....	1.5-1.6 lbs....	\$3.68-\$3.80 per 1,000.	\$0.38-\$0.77 per 100.
Basel, Switzerland.	Anthracite....				\$9.10 per ton.
Hamburg, Germany.	Bituminous....		About 6 lbs....	\$4.87 per ton....	\$7.50 per ton.
	Saxon brown coal.		About 6½ lbs....	\$4.04 per ton....	
Glasgow, Scotland.	Bituminous(?)	6 by 4 by 3½....		\$3.28-\$3.52 per ton.	\$4.25-\$4.38 per ton.
Roubaix, France...	Bituminous....	Cubes....			\$7.33 per ton.
		Large egg....			\$7.33 per ton.
Havre, France....	do....	Small egg....			\$7.72 per ton.
		Large....		\$5.40 per ton....	\$7.53 per ton.
Paris, France....	do....	Perforated....		\$5.60 per ton....	\$7.91 per ton.
		Egg....			\$9.65 per ton.
	do....	Perforated....			\$10.80 per ton.
		Solid....			\$9.26 per ton.
	Anthracite....	12 by 8 by 6....			\$4.60 per ton.
	do....	Egg....			\$5 per ton.
Frankfort, Germany.	Brown coal with 5 per cent coal tar and asphalt earth.	7 by 5 by 2....	1 lb....	\$2.90 per ton....	\$3.25 per ton.
	Bituminous....	12 by 8 by 6....	About 10 lbs....	\$4 per ton....	\$4.40 per ton.
Stettin, Germany...	Mixture of bituminous coal, brown coal, and tar.			\$2.93 per ton....	
	Brown coal....	6 by 2 by 1....	1.3 lbs....	\$4.05 per ton....	\$5.25 per ton.
Kehl, Germany....	Bituminous....	6 by 4 by 2½....	About 3 lbs....	\$4.52 per ton....	\$5.75 per ton.
	Brown coal....			\$3.85 per ton....	
Rotterdam, Holland.	Bituminous....			\$4.55 per ton....	\$5.27 per ton.
	Charcoal....			\$6.83 per ton....	
	Bituminous....			\$5.63 per ton....	
	Brown coal....			\$4.82 per ton....	\$5.43 per ton.

Nystrom ^a gives the labor cost of briquetting lignite for plants with one press as 45.4 cents per ton; two presses, 40 cents; three presses, 36.4 cents; four presses, 31 cents; five presses, 27.3 cents; and six presses, 22.3 cents per ton. He further states that "As the wages in Canada are higher than in Germany the cost of labor in a plant with one press is assumed at 60 cents per ton." He estimates the depreciation as 5 per cent per year and the maintenance as 3 per cent, and the cost of a complete plant with one press and a yearly capacity of 13,000 tons as \$75,000. His estimate of the charges for depreciation and maintenance is 8 per cent of \$75,000, or 46 cents per ton. The cost of labor and fixed charges, according to his estimate, is \$1.06 per ton.

In a consular report ^b the cost of briquetting lignite in Germany is stated as follows:

Cost of manufacture of lignite briquets.

Description.	United States currency.
From lignite taken from open workings under good conditions, with about 46 per cent water:	
In large briquet factories.....	\$1.14-\$1.29
In small briquet factories.....	1.19- 1.38
From lignite taken from open workings, with more than 46 per cent water (in large factories).....	1.38- 1.62
From lignite taken from deep workings, with about 46 per cent water:	
In large briquet factories.....	1.81- 1.62
In small briquet factories.....	1.62- 1.74
From lignite taken from deep workings, with more than 46 per cent water (in large factories).....	1.66- 1.86

In the case of materials containing 15 to 18 per cent of water, for which a drying process is not necessary, the cost of manufacture is considerably lower. As the cost of labor and fuel is higher in the United States than it is in Germany, the above figures are useful for comparison only.

Schorr ^c gives costs for the German lignite industry as follows:

The raw brown coal costs 22 to 31 cents per metric ton (2,204 pounds) at the works. The fixed charges, allowing 7 per cent for depreciation and 5 per cent for interest on the total investment, amount to about \$1.30 per metric ton of briquets. The wholesale price f. o. b. the works ranges from \$16 to \$23 per carload of 10 tons. The labor item rarely exceeds 24 cents per metric ton in small (one or two press) installations. All other manufacturing items, exclusive of fuel and fixed charges, are fully covered by from 8 to 11 cents per ton. In view of the relatively small output of these expensive presses, and in view also of the elaborate character of German briquetting works, the fixed charges (depreciation and interest) are high. The largest brown coal company owns 20 briquetting plants with 52 presses. There are a number of other concerns operating from 8 to 42 presses. Only a few factories employ less than 4 machines.

^a Nystrom, E., Peat and lignite; their manufacture and uses in Europe Bull. Canada Dept. of Mines, Mines Branch, 1908, p. 147.

^b Special Consular Report No. 26, 1908, p. 107.

^c Schorr, Robert, Lignite briquetting in Germany: Eng. and Min. Jour., vol. 85, 1908, pp. 460-461.

ESTIMATED COST OF BRIQUETTING AMERICAN FUELS WITH BINDERS.

The cost of briquetting fuels in the United States, using a binding material, varies according to the kind of binding material used and the locality in which the briquetting is done. For comparison of this method of briquetting lignite with that given in the previous estimate where no binder was used, the following data may be of interest:

Estimated cost of briquetting American fuels when a binder is used.

Item.	Per month.	Per ton briquets.
1. Fixed charges:		
(a) Interest on investment, 6 per cent on \$75,000.....	\$375.00	} \$0.179
(b) Depreciation—		
10 per cent on \$50,000, machinery.....	417.00	
5 per cent on \$25,000 (buildings, etc).....	104.00	
2. Labor:		
2 engineer-pressmen, at \$3.....	\$6.00	
2 firemen, at \$2.....	4.00	
2 binder men, at \$2.....	4.00	
6 laborers, at \$1.75.....	10.50	
12 men, total pay per day.....	24.50 X 25	
1 clerk or weigher, at \$50.....	50.00	} .133
3. Oil, waste, and light.....	50.00	
4. Repairs, maintenance, etc.....	100.00	.020
5. Binder 6 per cent (300 tons, at \$12).....	3,600.00	.342
Total briquetting cost.....		1.062

The above table is based on the following assumption:

Capacity of plant, 200 tons per day of 20 hours or 5,000 tons per month.

Plant operated 2 shifts of 10 hours each per day.

Cost of plant:

Machinery (American type)..... \$50,000.00

Building and tracks..... 25,000.00

Total..... 75,000.00

Cost of binder, per ton..... 12.00

Cost of fuel used for power to be included in cost of raw fuel.

The cost of briquetting lignite if a binder be used would, from the above estimate, be approximately as follows:

Cost of lignite briquets with binder.

Locality.	Value of raw fuel per ton.	Tons of raw lignite per ton briquets. ^a	Total cost of raw fuel.	Briquetting cost.	Cost of briquets per ton at plant.
Texas.....	\$0.87	2	\$1.74	\$1.06	\$2.80
North Dakota.....	1.49	2	2.98	1.06	4.04
California.....	1.64	2	3.28	1.06	4.34

^a Including fuel used for furnishing power as well as that used for making briquets.

The cost of briquetting bituminous, anthracite, or subbituminous coals would differ from the above estimate for lignite only in the

cost of binder required and the cost of the raw fuel. In the above estimate the briquetting cost, exclusive of binder, was \$0.342. If to this figure be added the cost of binder and the cost of fuel, a fair estimate of the cost of briquetting any of the three fuels will be obtained. The allowance to be made for fuel for power and drying will vary from 0.25 ton for anthracite, which is low in moisture and does not require drying, to 0.5 ton for subbituminous coal, which has considerable moisture (as high as 20 per cent).

The average briquetting cost at eight plants in the United States making briquets from anthracite dust or bituminous slack, including the cost of binder but not the fuel briquetted, was \$1.454.

Investigations have shown that certain classes of lignites can be briquetted without binder. They are superior to the raw fuel from which they were made, although not equal to bituminous and anthracite coals. Lignite briquets without binder do not hold together in the fire as well as those made with a binder, nor do they resist the effects of moisture as well. Briquets made without binder supply much of the household fuel in Germany. A plant making lignite into briquets without binder is obviously not dependent on a supply of binding material, which is an important consideration for a plant located in an out-of-the-way district. Briquets resist weathering much better than the raw lignite, have a higher heating value, are uniform in size, permit regular firing, and can be stored without serious deterioration. All lignites are not suitable for briquetting without binder and therefore those requiring a binder to agglomerate them are not considered in the above comparison.

It will require further investigation to determine whether Texas lignite can be briquetted commercially without binder. The results so far obtained indicate that binder will be required. North Dakota and California lignites, on the other hand, possess excellent briquetting qualities and probably can be briquetted without binder if a market be developed for this type of briquet. They can probably be made into better briquets by the addition of a binder that will make them stronger and more resistant to weathering.

With the present price of fuel in North Dakota, lignite briquets should there be made to compete with other fuels, but it is doubtful whether the same may be said of the Texas and California fields, where fuel oil and high-grade coal are available.

CHARACTERISTICS OF BRIQUETS.

COHERENCE.

Briquets should possess sufficient coherence to withstand handling and transportation. In a commercial plant briquets must withstand: (1) A drop from the machine to some form of conveyor; (2) the jar received while they are on the conveyor; and (3) a drop

from the conveyor to the pile of briquets already in the storage bins or cars. Fresh briquets are hot and soft and are much more easily broken than after cooling and air drying. Most modern plants are arranged so that the fresh briquets do not drop very far from the press to the conveyor belt, and in order to cool them quickly an open-mesh metal conveyor belt has been found most satisfactory, as it permits air to reach all sides of the briquets. The openings in the belt tend to prevent the briquets from rolling as on smooth rubber or leather belting. Scraper, screw, or bucket types of conveyors are not satisfactory for handling fresh briquets, as they break the briquets and do not allow air to come in contact with them. The drop from the end of the conveyor to the storage pile is lessened by the use of chutes.

After cooling and hardening the briquets should be sufficiently strong to stand whatever transportation is necessary to deliver them to the point of consumption with a dust and breakage loss not exceeding 5 per cent. The French use a tumbler test, similar to that described on pages 103 and 104, to determine the cohesive strength of briquets. The details of the test are as follows: ^a

One hundred briquets, each weighing 1.1 pounds (one-half kilogram), are placed in a cylinder 36.22 inches in diameter and 39.37 inches in length. The cylinder is divided into three compartments by diametrical partitions and revolves at a speed of 25 revolutions per minute. After having been charged, the cylinder is revolved for two minutes. The contents are then sifted upon a screen perforated with holes 1.12 inches square. The part that does not pass through this screen indicates the degree of cohesive force, which, in the case of the French Admiralty tests, should reach 52 per cent, or if the fuel be intended for torpedo-boat use, 58 per cent.

Briquets of any desired degree of coherence may be made by varying the proportion of binding material and the pressure. An increase of either the binder or the pressure increases the cost of manufacture. Experiments made by Wéry, of Paris,^a with a Biétrix machine may be taken as illustrative; the results are given below:

Effect on coherence of varying pressure and amount of binder.

Pressure in kilograms per square centimeter.	Pressure in pounds per square inch.	Pitch used.	Cohesion obtained.
		<i>Per cent.</i>	<i>Per cent.</i>
130	1,844	6	25
190	2,695	6	46
270	3,831	6	61
130	1,844	7	52
190	2,695	7	70
250	3,547	7	74

^a Briquets as fuel: Special Consular Report No. 26, 1903, p. 54.

HARDNESS AND TOUGHNESS.

The briquet should be hard, but not brittle. It is usually advantageous, therefore, to make the briquet of the minimum hardness that will suffice for the purpose in view. A briquet can be made harder by using a binder with a higher softening (melting) point. A larger percentage of the more brittle pitch is usually required.

The requirement of the French Admiralty is that the briquet should not soften at 60° C. (140° F.). Ordinarily it is sufficient that the briquet shall not soften on the hottest day.

DENSITY.

It is sometimes specified that the briquet should have a density of not less than 1.19. Perhaps a better standard would require the briquet to about equal in density the lump coal from which the slack is derived, this ranging from 1.1 to 1.4. The density is increased by pressure.

SIZE AND SHAPE.

The convenience of a briquet for a given purpose, and hence the extent of its use, will depend largely on its size and shape. Heavy rectangular blocks allow a large output for the investment and are consequently cheaper to manufacture, and more convenient to store. The French naval estimates show that 10 per cent more (by weight) of briquets can be stored in a given space than of lump coal, and the British Admiralty reports show a gain of as high as 20 per cent. The large rectangular briquets have the disadvantage of large smooth surfaces and are usually broken when fed into furnaces. To facilitate the breaking they are pressed with grooves or perforations. This gives better air circulation but decreases the output and the compactness of storage.

Prismatic briquets with rounded edges are the most popular abroad. Either these or ovoid shapes weighing less than 2 pounds are preferred for domestic use. The rounded edges cause much less dust and breakage on handling and insure good circulation of air and thorough combustion, but the briquets do not pack closely when stored and are somewhat harder to ignite.

WEATHERING.

The briquet should stand long exposure to the weather with little deterioration. A dense briquet will stand the weather better than a porous one. Briquets are liable to crack in the process of manufacture if they lack the proper proportion of binder, have been improperly mixed, are pressed too wet, or are insufficiently pressed. If the coal be finely ground, the briquets assume a more dense and polished surface, and are therefore more resistant to the weather.

Cracks allow the entrance of moisture and cause a rapid deterioration of the briquets on exposure to the weather. Lignite briquets do not stand long exposure to the weather as successfully as other briquets.

The binder used must be insoluble in water. The great obstacle to the successful use of starch, molasses, and sulphite-liquor residues as binders is their solubility, the cost of rendering the briquets water-proof being usually prohibitive. It is possible that in certain dry parts of the West the waterproofing of the briquets could be dispensed with altogether during the dry season, and to a considerable extent during the rainy season if the briquets were kept under cover. With pitches, tars, etc., a slightly increased percentage of binder is necessary in briquets that are to stand long exposure to the weather.

ABSORPTION.

The briquets should not absorb more than about 3 per cent of moisture. The proportion of moisture absorbed is increased when either the original fuel or the briquet is porous, or when the binder used has a tendency to be hygroscopic.

BURNING QUALITIES.

READINESS OF IGNITION.

The ease with which a briquet will ignite depends largely on the fuel used, but can be regulated to some extent. Large briquets ignite less readily than small ones. Sharp edges are an aid to ignition, but this advantage is not so great as to overcome the general preference for the prismatic and egg-shaped briquets. Briquets made from fine slack ignite less readily than those from coarser slack. A dense briquet is also more difficult to ignite. The use of an inorganic substance, such as clay or magnesia, as a binder, or as a constituent of the binder, tends to make the briquets ignite less readily. Increase of inorganic material, that is, ash, in the slack coal used produces the same result.

KIND OF FLAME.

The briquets should burn with a clear, intense flame and with little odor or smoke. The flame produced, as well as the smoke given off, will depend largely on the quality of the coal used and on the completeness of the combustion. The completeness of combustion can be regulated to some extent in the manufacture of briquets by making them of such a shape as to insure a good circulation of air and by the choice of a suitable binder. The smoke does not depend on the total amount of volatile matter in the briquets, but only on that part of the volatile matter that escapes before it is heated to the kindling temperature. The binder should not vola-

tilize before the temperature is sufficiently high to insure complete combustion of the gases formed.

Inorganic binders produce no smoke. Such organic binders as starch, molasses, or sulphite-liquor residues likewise do not volatilize until decomposed, and hence do not smoke, or smoke only little. Pitches, tars, and petroleum residues, when used as binders, volatilize, and will cause smoke and possibly odor if the gases formed are not completely burned. The regular shape of the briquets allows a better regulated air supply and enables a more complete combustion than does lump coal. This reduction of the smoke nuisance is one of the advantages to be derived from briquets.

RETENTION OF SHAPE.

Stability of form is important and depends on the properties of both the coal and the binder. The binder must hold the coal particles together until they are sufficiently softened to cohere. Some bituminous coals cake readily at a low temperature. Semianthracite coals follow next in order, and then anthracite coals. Some of the very hard anthracite coals with only a small proportion of volatile matter have little tendency to cake. Lignites as a class do not cake readily. Those from Oklahoma or New Mexico will cake sufficiently at a rather high temperature to hold together. Some California, Texas, and North Dakota lignites show practically no tendency to soften or cake at any temperature. With such lignites it is extremely difficult to make a briquet that will retain its shape in the fire. Briquets for domestic use can be made from such lignites. These briquets, with the use of a suitable grate, might be used in a variety of manufacturing operations. As a locomotive fuel they would be less suitable.

With a readily caking coal, a binder that volatilizes at a comparatively low temperature may be used. With coals that cake at higher temperatures a less volatile binder must be used. With a lignite that does not cake, the only binder that will enable the briquets to retain their shape until completely consumed is an inorganic binder that does not volatilize, unless sufficient organic binder be added to practically coke the briquets. With such lignites, organic binders that do not volatilize, such as starch, molasses, and sulphite-liquor residues, give results that are fairly satisfactory. The briquets retain their shape until the binder is itself decomposed. As the inorganic binders add ash and as the other nonvolatile binders mentioned are not waterproof, it may be possible to mix a noncaking coal with a sufficient proportion of caking coal. Then with a suitable binder the briquets will retain their coherence in the fire by the softening of the caking coal used.

PERCENTAGE OF ASH.

The proportion of ash left when the briquets are burned is the sum of that contained in the slack and in the binder used. In some foreign countries only 6 per cent of ash is permitted under many of the contracts for briquets. When the ash content of the slack exceeds 6 per cent the briquet may be improved by washing the slack coal before briquetting. This saves freight on an incombustible material, saves binder, and gives in every way a better and more concentrated fuel.

EVAPORATION RESULTS.

The theoretical heating value of a briquet is the sum of the heating units of the coal and of the binder. Organic binders usually equal or exceed in heating value, weight for weight, the slack coal used. They increase the total heat in a given weight of fuel, but owing to the small percentage of binder added, this increase is slight. But the briquets have the advantage over the coal in that they burn with less waste and permit easy regulation and more complete combustion. The evaporation results should at least equal those of the best lump coal from the screenings and dust of which the briquets are made. In many cases the efficiency of the fuel may be increased 10 to 15 per cent by briquetting, when special furnaces are not provided for burning the fuel in its natural condition.

EQUIPMENT OF BRIQUETTING PLANTS.**ST. LOUIS PLANT.**

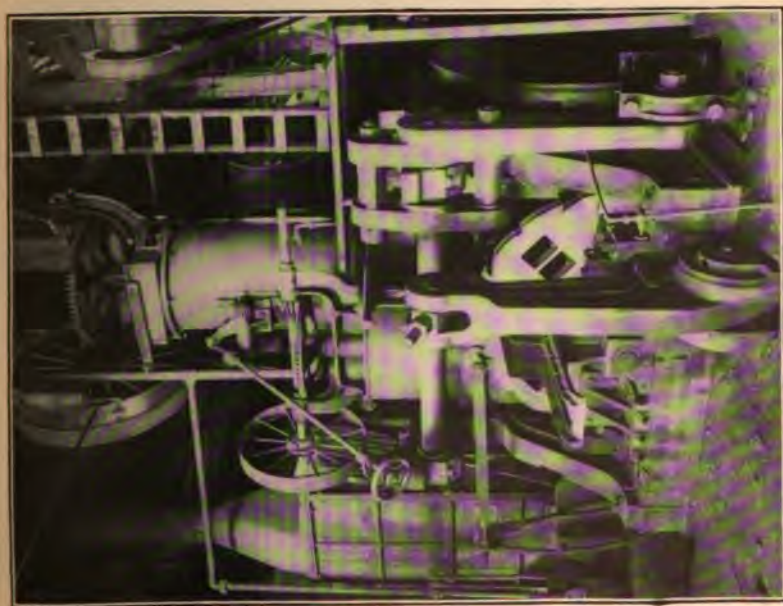
The briquetting plant erected at St. Louis in 1904 had two machines of distinctly different types, one of English and one of American manufacture. The English machine is one of the standard type for briquetting with stiff pitch, and consists of a double-compression, vertical-table press, with closed molds, and accessories. It was operated by a 50-horsepower Westinghouse motor.

The American machine was constructed on Belgian patterns, being what is ordinarily known as the eggette, or Belgian, type of press. It, too, was operated by a 50-horsepower Westinghouse motor.

ENGLISH MACHINE.

The English machine as installed at St. Louis is illustrated in Plate I, A, which is a general view of the machine.

The coal was delivered to the feeding platform of the English machine by a Robins inclined belt conveyor. It was then shoveled to a coal-feeding worm through a hole in the floor. The pitch that was used as a binder was first reduced to suitable lumps and fed into the pitch cracker, where it was broken to $\frac{1}{4}$ -inch size and dropped upon



4. ENGLISH BRIQUETTING MACHINE AS INSTALLED AT ST. LOUIS.



B. SCHLICKEYSEN PEAT MACHINE.

a smaller worm; this in turn delivered it to the coal in the larger worm. The smaller worm was driven by sprockets from the larger one, and the percentage of pitch that was added to the coal could be varied by changing the wheels.

Although this arrangement gave good results when the same coal and binder were used, it was not sufficiently flexible to be adapted to the various coals and binders that were used in the experimental work. For most of the experiments, the crushed coal and cracked binder were weighed and mixed by hand. The mixed coal and binder were then fed to the larger worm through the hole in the floor.

After a slight mixing in the larger worm the material entered an impact disintegrator, in which it was reduced to the desired fineness. The speed of the disintegrator was varied according to the character of the coal. From the disintegrator the material was elevated to a pug mill in which the binder was softened by live steam. As there was no superheater, the range of temperature in which the binder could be softened was limited. Again, live steam is apt to be wet, and was therefore often a detriment to the production of successful briquets. A Foster superheater that was afterwards installed overcame these difficulties almost entirely.

From the pug mill the plastic mixture fell by gravity to the press feed box, from which it was forced by a plunger into brass-lined molds on a vertical revolving table. After a half revolution of the table, the mass in the mold was pressed by a system of compound levers at a pressure of 2 tons per square inch. The pressure, however, could be varied from a few pounds to 4,000 pounds per square inch.

After another quarter revolution, by means of a plunger, the briquets were forced out of the molds to a slide or table set at a height convenient for stacking on trucks. The briquets were taken from the table by hand as soon as they were discharged from the press and stacked on a platform just outside the building. The briquets were rectangular (6 by 5 by 4 inches) except for rounded corners, and weighed on the average 6.8 pounds each. The maximum capacity of this press was 6 tons of briquets per hour.

To obtain a briquet that would more nearly fulfill the requirements of domestic use and of stationary and locomotive boiler practice, the thickness of the briquet was reduced to $2\frac{1}{2}$ inches, the other dimensions remaining the same. By adjusting the machine for this change it was possible to make satisfactory briquets, weighing about $3\frac{1}{2}$ pounds, at the rate of about 4 tons per hour.

This machine was adapted only to those binders that did not become plastic before reaching the pug mill. With inorganic binders or binders made of a number of different materials, it was found neces-

sary to weigh and thoroughly mix the coal with these binders before feeding the materials to the machine. This practice assured a definite percentage of binding material to the quantity of coal taken.

In order to obtain a greater range in the size of the crushed fuel for the briquets, in 1905 a series of pulleys was installed on the driving and driven shafts operating the disintegrator, giving approximately 80, 65, and 50 per cent of its original speed. By these improvements easier and more continuous operation was obtained.

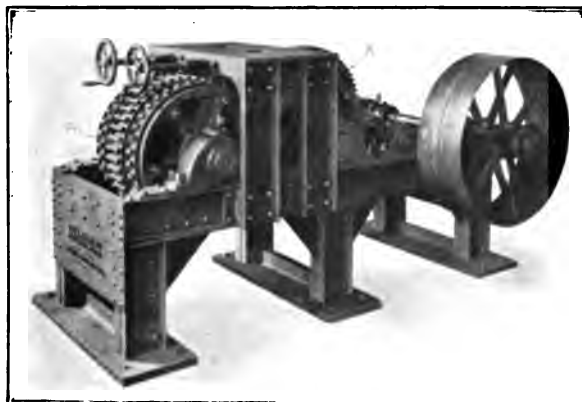
A fire occurred in the latter part of 1905 and destroyed the briquetting plant and all samples of briquets and fuels stored in it. The English machine, however, was not seriously damaged by the fire, but in repairing it considerable care was taken to strengthen certain parts, to provide better lubrication and alignment, and to replace the tight and loose pulleys with friction clutches. The new plant was first operated in May, 1906, and as a result of previous experience it contained some improvements in the methods of crushing, mixing, and conveying the fuel, and of preparing the fuel and binder before briquetting. The new auxiliary equipment for the two machines was not materially different from that originally belonging to the English machine.

The foundation of the Stedman disintegrator was built up level with the floor so as to be more accessible. A 40-ton storage bin was built above the charging floor to allow coal to run by gravity into the hopper scales placed on this floor. A pitch cracker was so placed beside the scales that pitch from it and coal from the scales could pass uniformly to a mixing conveyor directly under the charging floor and be discharged into the disintegrator on the floor below.

AMERICAN MACHINE.

In operating the American plant the coal was received from the Robins conveyor at the level of the second floor in a 3-ton bin. The coal was drawn from the bin to a Fairbanks platform scales, from which it passed into the boot of an elevator, from which it was delivered into a 15½ cubic foot measuring box.

The coal was then dumped into a Buffalo mixer, fitted with steam jackets. The binder was melted in a small steam-jacketed tank that was entirely distinct from the main part of the machine. From the tank it was dipped by hand in the desired proportion and poured into the mixer. After a thorough mixing the mass was dropped into the feed slides of a tangential press. This press consisted of two pairs of narrow-faced rolls (A, Pl. II, *A* and *B*), in the tires of which were ovoid cup-shaped molds. As these rolls revolved, the excess material was squeezed out, the pressure used depending upon the viscosity of the mixture that was being compressed. The 5-ounce eggettes were delivered from under the press to a short rubber belt (B, Pl. II, *B*).



A. AMERICAN BRIQUETTING MACHINE AT ST. LOUIS.



B. AMERICAN BRIQUETTING MACHINE AT WORK.

The capacity of the press was 5 tons per hour, but the output of the mixing device was much too small for operating the press at this capacity. The press had one advantage over the larger English plant, inasmuch as it permitted the testing of mixtures in lots as small as 15 pounds, these small quantities being thrown directly into the feed hopper of the press. In making the small tests and those in which it was desired to use a stiff binder, a 35-gallon kettle was employed for heating the mixture. It was also possible to test on the American machine the mixtures used in the English machine by crushing some of the hot briquets as soon as they were received from the molds and transferring them at once to the feed box of the American machine.

RENFROW NO. 1 MACHINE.

In rebuilding the plant at St. Louis after the fire, an experimental briquetting machine of the plunger type, known as the Renfrow No. 1 machine (Pl. III), was installed. The main difference between it and the English machine was that the Renfrow machine made briquets at each end of the stroke of the plunger. By this design the briquets were kept under compression in the dies twice as long for the same output as they were in machines of the English type. The same length of time for compression being assumed, however, and other conditions being equal, the Renfrow machine had double the output of other plunger-type machines. The period of compression was determined by experience and fixed the speed of the machine. The longer the pressure remained on the warm charge the greater the cohesion and the better the briquet.

Twelve briquets were made on the Renfrow machine at each end of the stroke, or 24 in each revolution. At 12 revolutions per minute the output was 4 tons per hour. The pressure obtainable under ordinary working conditions never exceeded 1,000 pounds per square inch, whereas a pressure of 2,000 to 2,500 pounds per square inch is considered necessary to make satisfactory briquets. For this reason it was found necessary to use a softer pitch and a greater percentage of binder than on the English machine. The fault was somewhat obviated by heating with steam injected directly into the mixture.

A bucket elevator and a divided chute provided with a gate served to convey the fuel from the disintegrator either to the agglomerating cylinder of the English machine or to the storage bin above and directly behind the Renfrow machine. This bin contained an agitator to prevent the fine fuel from packing. The fuel was carried from the bin to the hoppers of the Renfrow machine by belt conveyors. Provision was made for the use of either liquid or hard pitch as a binder for briquets made on the Renfrow machine. During the hot weather considerable difficulty was experienced in conveying the finely divided and thoroughly mixed coal and pitch from the dis-

integrator to the Renfrow machine. This difficulty was obviated by introducing the melted hard pitch into the coal as it entered the hoppers of the machine, care being taken to keep uniform the temperature and rate of flow of the pitch.

The necessity of maintaining a comparatively large force of laborers about the plant for careful handling of samples, both in the raw and briquetted form, justified the use of more labor in the actual briquetting operations than would be found in a commercial plant. The thorough cleaning of all conveyors, bins, and auxiliary apparatus before briquetting each sample was of as much importance for comparative results as the actual briquetting of the fuel; consequently no attempt was made to develop mechanical means for doing this work.

The briquets made by the Renfrow No. 1 machine were biscuit shaped, being short cylinders about 3 inches in diameter with convex ends. Their extreme length varied from $1\frac{1}{2}$ to 3 inches according to the fuel. The average weight of the briquet made by this machine was 8 ounces. At the close of the tests at St. Louis, in March, 1907, the machine was returned to its owner.

HAND PRESS.

The experimental laboratory contained a small hand press designed for a pressure varying from 0 to 5,000 pounds per square inch. The briquets were cylindrical, one-half inch in diameter, and could be made of any length from one-fourth inch to $1\frac{1}{4}$ inches. The results obtained on this hand press were verified by the briquets made on the large machines, the formulas for the mixtures having been derived from experiments made with the small press. This press should not be confused with the hydraulic hand press used in experiments at Pittsburgh, described on page 34.

NORFOLK PLANT.

GENERAL DESCRIPTION.

The equipment of the United States Geological Survey briquetting plant at Norfolk comprised three briquetting machines; heating and mixing apparatus; storage bins for the raw fuel; crushers, grinder, and disintegrator for reducing the fuel to the desired fineness; machines for crushing or "cracking" pitch; scales; and the necessary elevators and conveyors. Most of the equipment had been used for varying periods at the St. Louis plant, but of the three machines installed the English machine was the only one that had been used at the St. Louis plant. The machines were (a) the English or Johnson press; (b) the Renfrow No. 2 machine; and (c) a Schlick-eyen peat machine.



GENERAL VIEW OF BRIQUETTING MACHINES, ST. LOUIS PLANT.

The general plan of the plant is shown in Plate IV. There were two 25-ton storage bins behind the English machine and one 50-ton bin behind the Renfrow No. 2 machine. The fuel was delivered from the pit at the car track by a bucket elevator and a 16-inch belt conveyor to the storage bins.

Under the coal-storage bins and behind the briquetting presses was the "pitch-mixing platform." On this platform were scales, to which coal could be drawn from the storage bins and weighed. Back of the English machine were other scales to weigh the pitch used as a binder. The Renfrow No. 2 machine had an adjustable mechanical device for keeping the ratio of pitch to coal constant, so that it was not necessary to weigh the pitch used with this machine. A 30-inch Jeffrey belt conveyor 33 feet long (Pls. IV, V, A) carried the briquets from the front of the machines to the outside of the building, and a 24-inch Robbins belt conveyor 65 feet long (Pls. IV, V, A) carried the briquets from the first conveyor and loaded them into the railroad cars. Plate V, B, shows the delivery end of this conveyor.

ENGLISH MACHINE.

The English machine was the same that was erected and used at the St. Louis plant at the time of the Louisiana Purchase Exposition. This machine is described on pages 22 to 24 of this report. As installed at Norfolk, the capacity of this press was 3.8 tons per hour.

The position of the English machine with reference to the Renfrow No. 2 machine and the coal-grinding equipment at the Norfolk plant is shown in Plate IV. Plate VI gives further details of arrangement and equipment.

The scales back of the English machine (Pls. VI, VII, A) were placed about 6 inches above the floor to allow room for a square wooden plunger sheathed with sheet iron (D, Pl. VI) to push the weighed fuel into a hole in the floor in front.

Beneath the floor was a horizontal worm conveyor (E, Pl. VI) to carry the fuel and pitch from the scales to a chute leading to the Stedman disintegrator (F, Pl. VI and Pl. VII, B). Suspended in this chute was a powerful electromagnet intended to pick up any pieces of iron or steel that might be in the stream of material, and to prevent them from causing damage to the disintegrator or the press. The two sets of hammers in the disintegrator revolved in opposite directions at 790 revolutions per minute. This disintegrator stood on a concrete foundation, and its base was 2 feet above ground level.

In front of the disintegrator a pit 5 feet deep, $3\frac{1}{2}$ feet wide, and 5 feet long, walled in and floored with concrete, was provided for the boot of a bucket elevator (H, Pl. VI). The elevator lifted the ground fuel and pitch to a point above and back of the briquetting machine, whence a covered wooden chute lined with sheet iron

(I, Pl. VI) led to the upper mixing cylinder (J, Pl. VI) of the machine (Q, Pl. VI).

At the base of the upper mixing cylinder steam could be admitted through several openings. Water was supplied at the top of this cylinder when required. Below the mixing cylinder was the feeding cylinder (K, Pl. VI) of the briquetting press.

To conduct a briquetting test on the English machine, coal was drawn from one of the storage bins, weighed, and mixed with binder.

After the fuel and the binder had been thoroughly mixed and properly heated the mass became more or less plastic, according to its briquetting qualities, and was drawn into the feed box through a door in the lower part of the cylinder. From the feed box the mixture was forced by a plunger into the molds in the vertical revolving table of the press. The plunger speed was 17 strokes per minute, 1 stroke for each revolution. At half a revolution the mass in the mold was pressed by a system of levers from each end, the maximum pressure being about 4,000 pounds per square inch. Two briquets were formed at each stroke, or 34 briquets per minute.

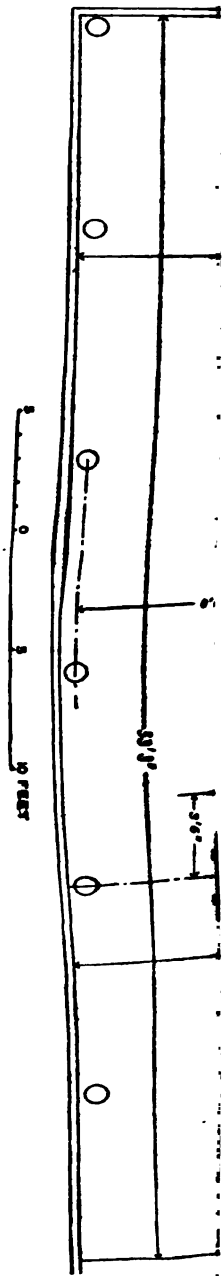
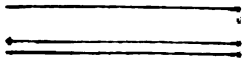
The finished briquets were removed from the machine by hand and either stacked near the machine or placed on the conveyor in front of the Renfrow No. 2 machine and by it loaded directly into a car. They were rectangular, with rounded corners, $6\frac{1}{2}$ inches long, $4\frac{1}{2}$ inches wide, and $2\frac{1}{2}$ inches thick. Their average weight was about $3\frac{1}{2}$ pounds. The maximum capacity of the machine was about 30 tons per 8-hour day.

RENFROW NO. 2 MACHINE.

The Renfrow No. 2 machine installed at Norfolk was an improvement over the No. 1 machine, as defects of the first machine, brought out by the tests, were corrected. The new machine had a stronger frame to permit the use of heavier pressure. The working parts not subjected to pressure were of bronze to prevent corrosion; the dies, cams, and rollers were of tool steel. Provision was made in the design of the housing for the removal of any working part without dismantling the machine, and the arrangement of the die plunger and the length of spring behind the plunger were such as to reduce to a minimum the chance of a double charge entering the press and to prevent damage to the machine if one did enter.

The Renfrow No. 2 briquets were cylindrical, with convex ends. They measured $3\frac{1}{2}$ by $1\frac{1}{2}$ inches and weighed 11 ounces. This machine was a closed-mold double-acting plunger machine, forming 12 briquets at the end of each stroke, or 24 at each revolution. Running at 9 revolutions, or 18 strokes per minute, the capacity of the machine was about 8.9 tons per hour.

BUREAU OF MINES

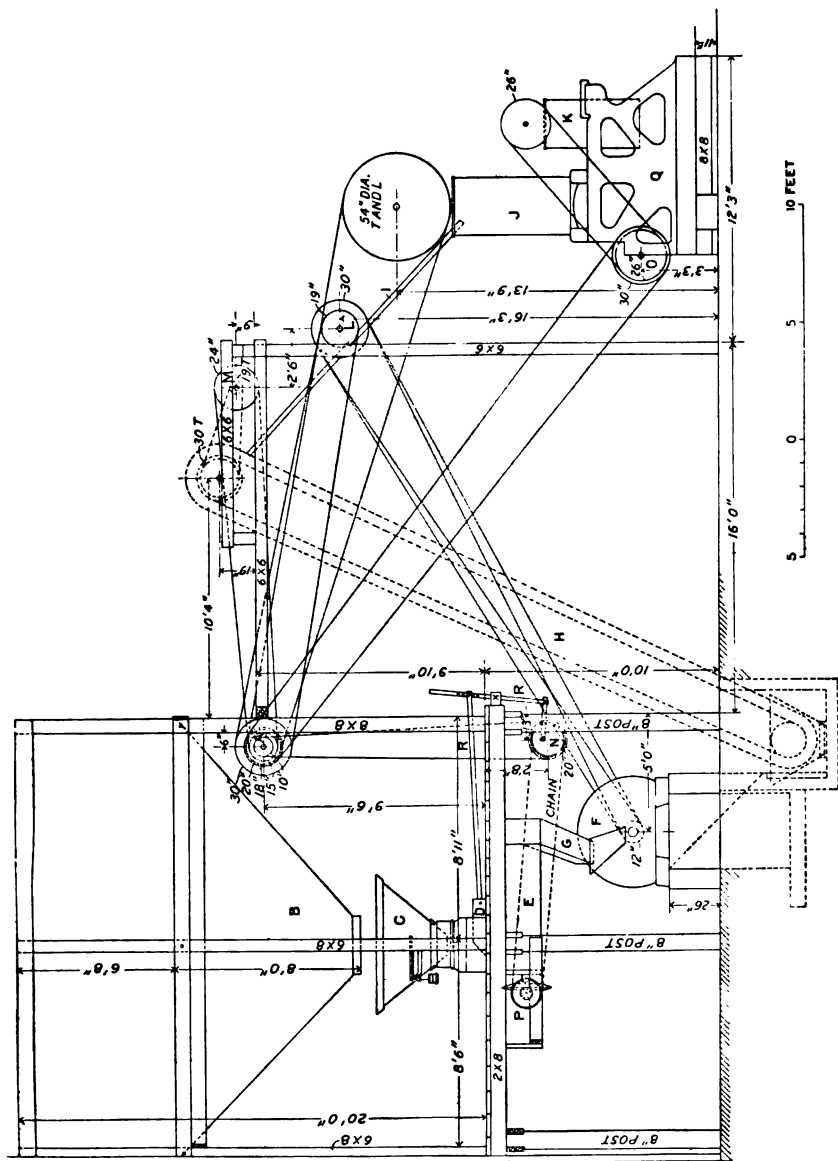




A. PLANT END OF CAR-LOADING CONVEYOR, SHOWING END OF CROSS CONVEYOR.



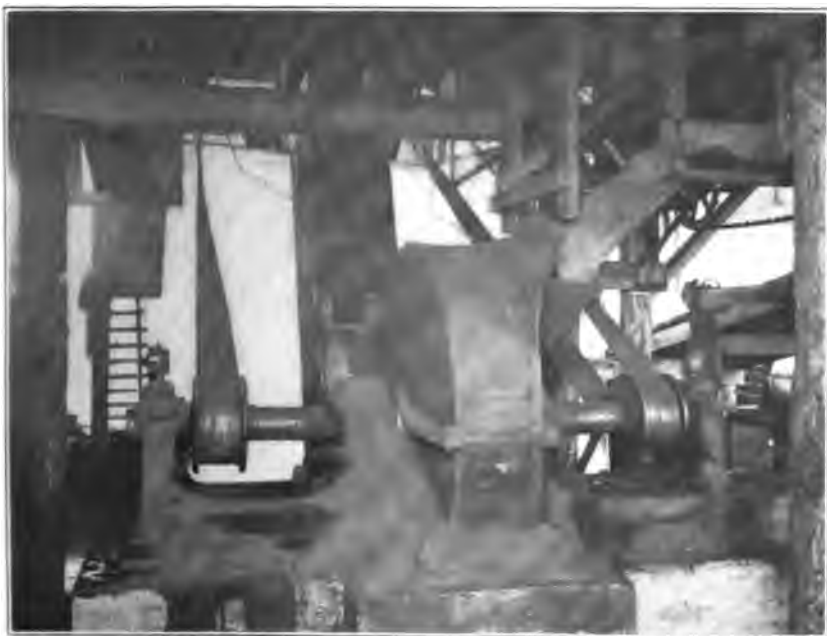
B. CAR-LOADING CONVEYOR FILLING CAR WITH BRIQUETS.



ELEVATION OF ENGLISH BRIQUETTING MACHINE.



A. HOPPER AND SCALES BACK OF ENGLISH MACHINE.



B. STEDMAN DISINTEGRATOR.

The location of the storage bin, mixing platform, coal crusher, conveyor, the pulleys and belting that drove the press, and the various accessories are shown in Plate VIII; the location of the machine with reference to the English machine is shown in Plate IV.

The Renfrow No. 2 machine equipment included a 50-ton bin (H, Pl. VIII), situated above and behind it; hopper scales (I, Pl. VIII and Pl. IX, A); a pitch cracker (L, Pl. VIII, and Pl. IX, A); a hopper with adjustable slides in the bottom at the coal scales (J, Pl. VIII); an apron conveyor for the cracked pitch (M, Pl. VIII); a Williams mill (K, Pl. VIII); and a bucket elevator (N, Pl. VIII). The machine itself included steam-jacketed cylinders (PPP, Pl. VIII) for heating and mixing the fuel and binder; worm conveyors for forcing the mixture through the cylinders; a feed box (T, Pl. VIII); a die filler (U, Pl. VIII); a conveyor for bringing the briquets to the front of the machine, and a double-acting press (R, Pl. VIII):

For a test on the Renfrow machine sufficient fuel to fill the hopper on the scales (Pl. IX, A) was drawn from the storage bin and weighed; the slide at the bottom of the scales was opened, and the fuel passed through a hole in the floor of the pitch platform into a hopper over the apron conveyor. A sliding door on the side of the hopper could be raised or lowered to vary the thickness of the layer of fuel on the conveyor.

Pitch broken by hand into lumps was fed into the pitch cracker (Pl. IX, A). The apron conveyor under the pitch hopper discharged the pitch in an even layer upon the fuel lying on the fuel conveyor below. Plate IX, B, shows the arrangement of these conveyors. The fuel and pitch were ground in the Williams mill and elevated to the hopper above the Renfrow No. 2 machine. The screw conveyors moved the material flowing from the hopper through the three pairs of horizontal steam-jacketed drums, in which the fuel and binder were suitably heated and thoroughly mixed. The briquet mixture then passed to the press box, whence plungers forced it into die fillers. Then the briquets were formed under an average pressure of 1,000 pounds per square inch. The briquets were discharged from the molds on the return stroke. The capacity of the Renfrow No. 2 machine was 71 tons per 8-hour day when making 18 strokes per minute.

SCHLICKEYSEN PEAT MACHINE.

The Schlickeysen peat machine used at the Norfolk plant for experimenting with North Carolina peat was of the auger type. The peat was macerated by a screw or worm and then squeezed through a tapered tube in a continuous stream; briquets of the proper length were cut off by a wire cutter. The general appearance of the machine is shown in Plate I, B (p. 22). In addition to the machine itself,

the equipment consisted of a scraper conveyor for elevating the peat to the top of the machine and a number of wooden shelves, about 2 by 4 feet and 4 inches high, which were nailed to cleats on edge. The shelves received the briquets direct from the press and could be stored in tiers one above the other to permit drying.

PITTSBURGH PLANT.

GENERAL DESCRIPTION.

At the experiment station of the Bureau of Mines at Pittsburgh a complete lignite-briquetting plant is installed in the main part of the briquetting building. The equipment, which was furnished by the *Maschinenfabrik Buckau Actien Gesellschaft*, Magdeburg, Germany, is as follows: Tubular drier, sorting sieve, auxiliary crushing rolls, cooler, screw conveyor, large storage hopper, briquetting press directly connected to a steam engine, apparatus for conveying briquets, shafting, hangers, and extra sets of dies. To complete the plant the United States Geological Survey erected a coal elevator, a crushing roll, and the necessary belting and piping, in addition to the building and stacks.

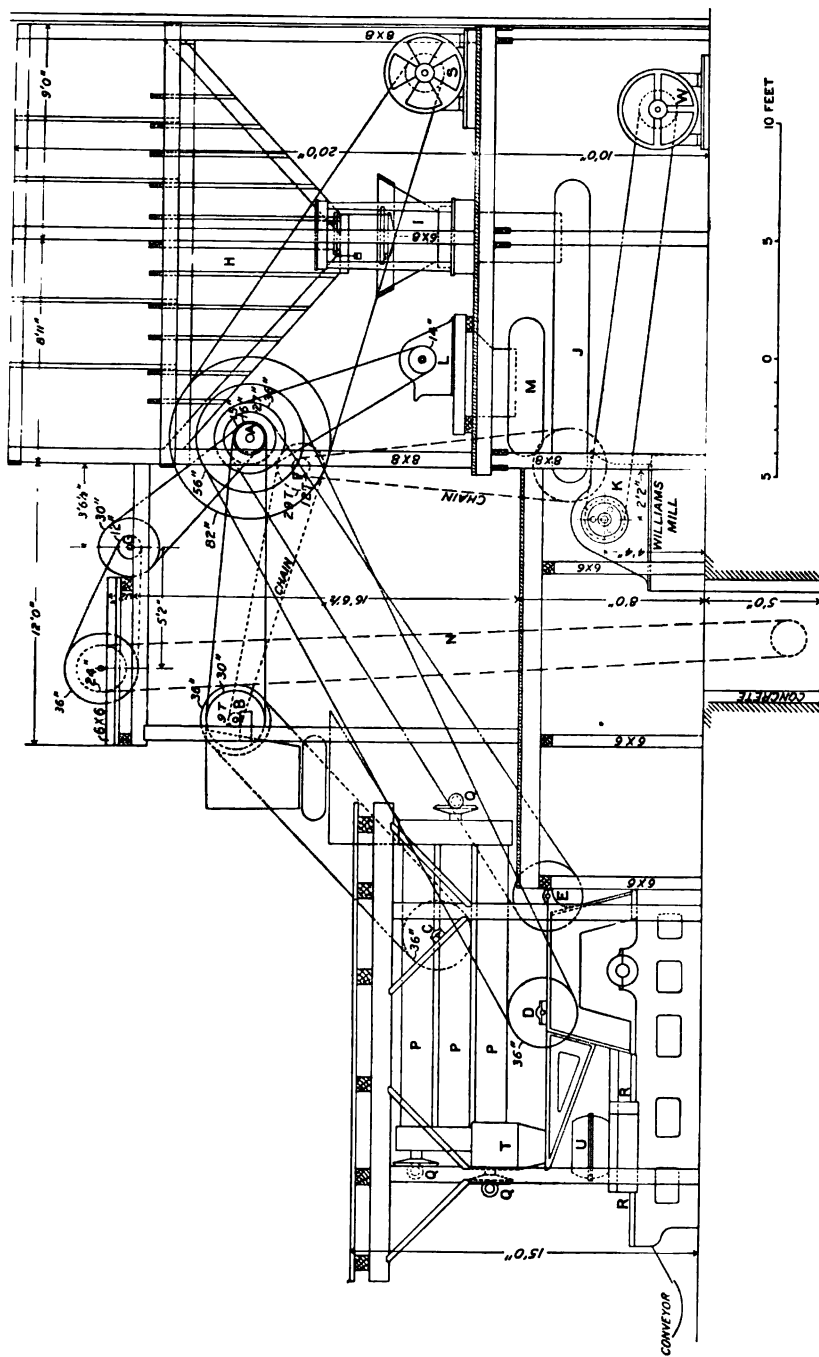
The English machine used at St. Louis and Norfolk for briquetting anthracite and bituminous slack and coke breeze was set up in the building, but no provision has been made for using it to briquet large quantities of fuels. Work at this plant has been concentrated on tests of lignite with the German machine.

Briquetting machines at Pittsburgh and their equipment are housed in a steel-frame building inclosed with curtain walls of reinforced concrete 2 inches thick. The steel work is of heavy construction, for it has to carry not only the machinery but also, on the third floor, a storage room to hold 100 tons of coal or lignite. Plate X illustrates the general appearance of the building.

CONVEYOR AND CRUSHER.

Two 60-ton bins for storing the raw lignite were built near the car track at one side of the briquetting building, so that the lignite could be shoveled directly into them from the box cars in which it had been shipped from the mine. A bucket elevator (fig. 1) of the link-belt type carries the raw lignite from the receiving bin to the third floor of the building.

The corrugated single-roll crusher (A, figs. 1 and 2) installed on the third floor was expected to break run-of-mine lignite into pieces three-eighths of an inch in diameter or smaller at one operation. The North Dakota lignite, being tough and woody, was especially hard to crush, and some samples had to be run through the crusher several times. The tests indicated that lignite should be crushed in stages,



ELEVATION OF RENFROW NO. 2 BRIQUETTING MACHINE.



A. PITCH CRACKER, SCALES, AND HOPPER BACK OF RENFROW NO. 2 MACHINE AT NORFOLK.



B. ARRANGEMENT AND DRIVING GEAR OF PITCH AND COAL MIXING CONVEYORS.



BRIQUETTING-PLANT BUILDING AT PITTSBURGH.

first by a toothed roll and then by a smooth or corrugated roll or by a disintegrator. In Germany three sets of crushers are used in lignite-briquetting plants.

DRIER.

The Schulz tubular drier (C, figs. 1 and 2), on the second floor of the building, is similar to an ordinary multitubular boiler, 22½ feet long, 7½ feet in diameter, and contains 195 tubes of 3¼ inches inside diameter, evenly spaced around the drum. The material to be dried passes

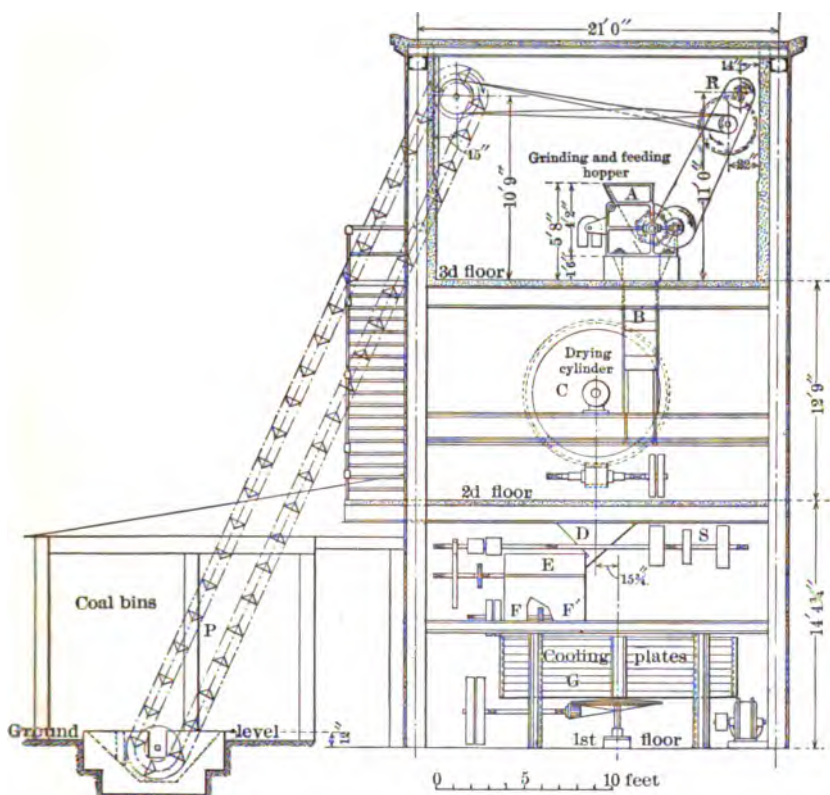


FIGURE 1.—Cross section of lignite-briquetting plant.

through the tubes, which have a drying surface of 4,310 square feet. A worm gear turns the drier at a speed of 5.2 revolutions per minute.

A 5-inch pipe carries the exhaust steam from the engine of the briquetting machine to the drier, the steam entering the latter through the upper bearing. During the tests steam pressures of 3 to 30 pounds per square inch are used, according to the degree of dryness desired. Live steam is used when the engine is not running. The ground lignite is fed into the drier tubes by a chute (Pl. XI, A) from the roll crusher.

The lower or delivery end of the drier is connected with a revolving screen by a hopper (D, figs. 1 and 2, and Pl. XI, B). This screen has openings of two sizes; those in the receiving half are three-sixteenths inch square, and those in the other half are three-eighths inch square. Material larger than three-eighths inch is discharged from the end of the screen through a waste pipe.

The material that passes the $\frac{1}{8}$ -inch openings falls through a chute (F', fig. 1) into the cooler, and the material passing the three-eighths-

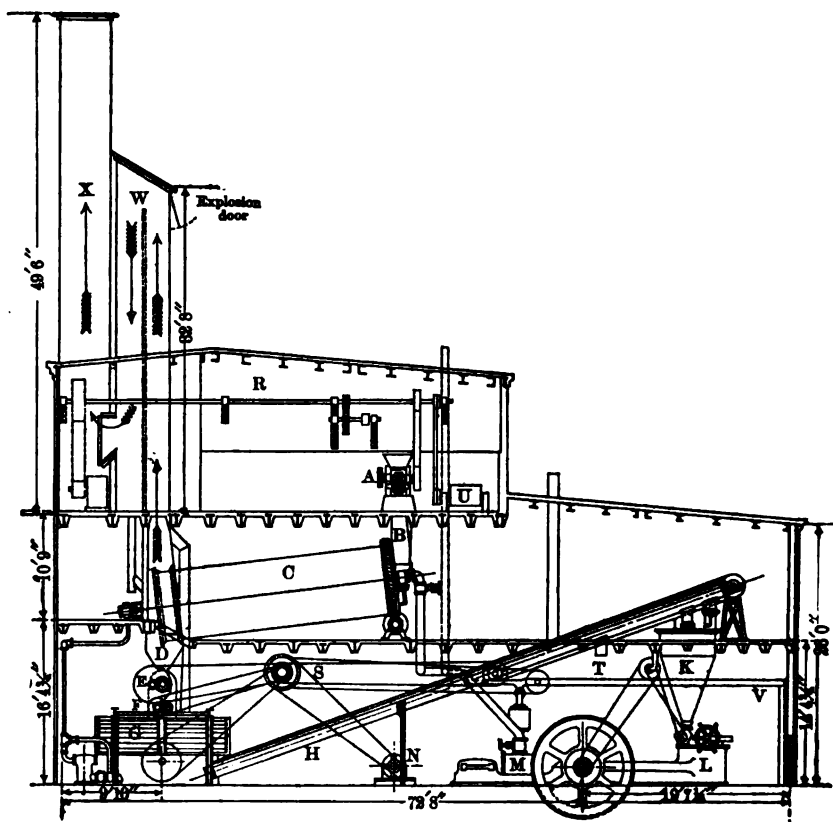


FIGURE 2.—Longitudinal section of lignite-briquetting plant.

inch screen falls on a pair of rolls, is crushed to a fine powder, and then goes into the cooler shown at F, figure 1.

COOLER.

The cooler (G, figs. 1 and 2, and Pl. XII, A) has four circular stationary plates 13 feet in diameter, arranged one above the other, with raised edges and closing plates to keep the dust from escaping. Over each plate four radial arms, each carrying several scrapers set at an angle of 45° with the arms, are revolved by a vertical central



A. RECEIVING END OF SCHULZ DRIER.



B. DELIVERY END OF SCHULZ DRIER.



1. SCREEN, CRUSHING ROLLS, COOLER, AND ELEVATOR TO SECOND FLOOR OF BUILDING.



2. LABORATORY HAND PRESS

shaft. The material is scraped from the outer edge of the upper plate toward the center, where it falls through holes to the plate below; another set of scrapers moves it to the outer edge of that plate and another set of holes. Thus it reaches the bottom plate, from which it goes to a worm conveyor (H, fig. 2, and Pl. XII, A). This conveyor elevates the cooled material to the second floor, where it falls through chute I (fig. 2) into large hopper K (fig. 2) over the German press L (fig. 2).

DUST STACKS.

The dust and gases from drying lignite may form explosive mixtures with air; hence stacks (W and X, fig. 2) were erected over the delivery end of the drier to carry off the dust and gases, as shown by the arrows in figure 2. An explosion door is placed at the upper end of stack W. Most of the dust settles in the down-coming section of stack W and goes to hopper D through a chute. A spray was put in stack X to wash out any dust that remained in the gases and to condense steam and other vapors. The air current up the stacks tends to increase the capacity of the drier.

GERMAN MACHINE.

The German machine (Pl. XIII) was designed to briquet either peat or lignite that will cohere under pressure because of inherent bituminous matter. Attempts to use a similar press for briquetting materials containing artificial binder will probably result in a stalled machine or a broken stamp.

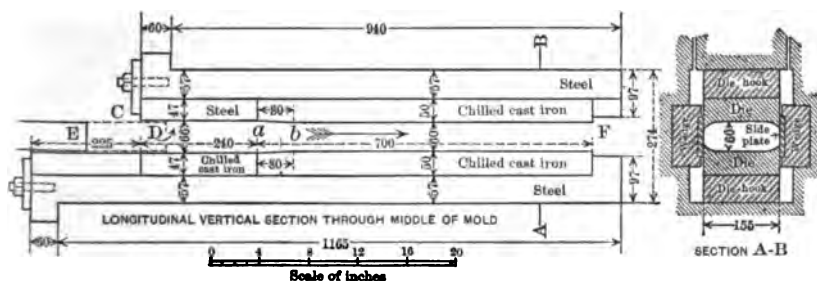


FIGURE 3.—Vertical sections through mold box of lignite-briquetting press; dimensions are in millimeters (25.4 mm.=1 inch).

The machine is of the open-mold type. As the material passes through the mold it is reduced in volume, because the opening in the delivery end of the mold is smaller than the one in the receiving end. This difference in the size of the two openings is shown by the vertical sections through the mold given in figure 3. In the following paragraph the letter references pertain to this figure.

A charge enters the mold at C, is pressed by the stamp E into the end of the mold and is pushed along to D, the end of the travel of the stamp E. The stamp moves back and then presses another charge to D. The first charge is forced along under heavy pressure in the direction of the arrow until it reaches *a*; as it passes from *a* to *b*, in what is called the die angle, it is compressed at top and bottom by the reduction in height of the mold. From *b* the briquet is forced along by the successive charges of material and its sides are hardened and polished. When it leaves the mold at F the briquet is hot, but after it has traveled a few feet in a trough it is ready for storage or loading into cars.

The briquets made by the German press have flat sides and rounded ends. A good idea of their appearance may be obtained by referring to Plate XVI. They measure approximately $6\frac{1}{2}$ by $2\frac{1}{2}$ by 1 inch and weigh about 1 pound apiece.

ENGLISH MACHINE.

The English machine, shown in Plate XIV, was described on pages 27 to 28. This machine is suited for briquetting any solid fuel with added binder. No provision was made for experiments with this machine during the fiscal year ending June 30, 1909, and its equipment was not completed.

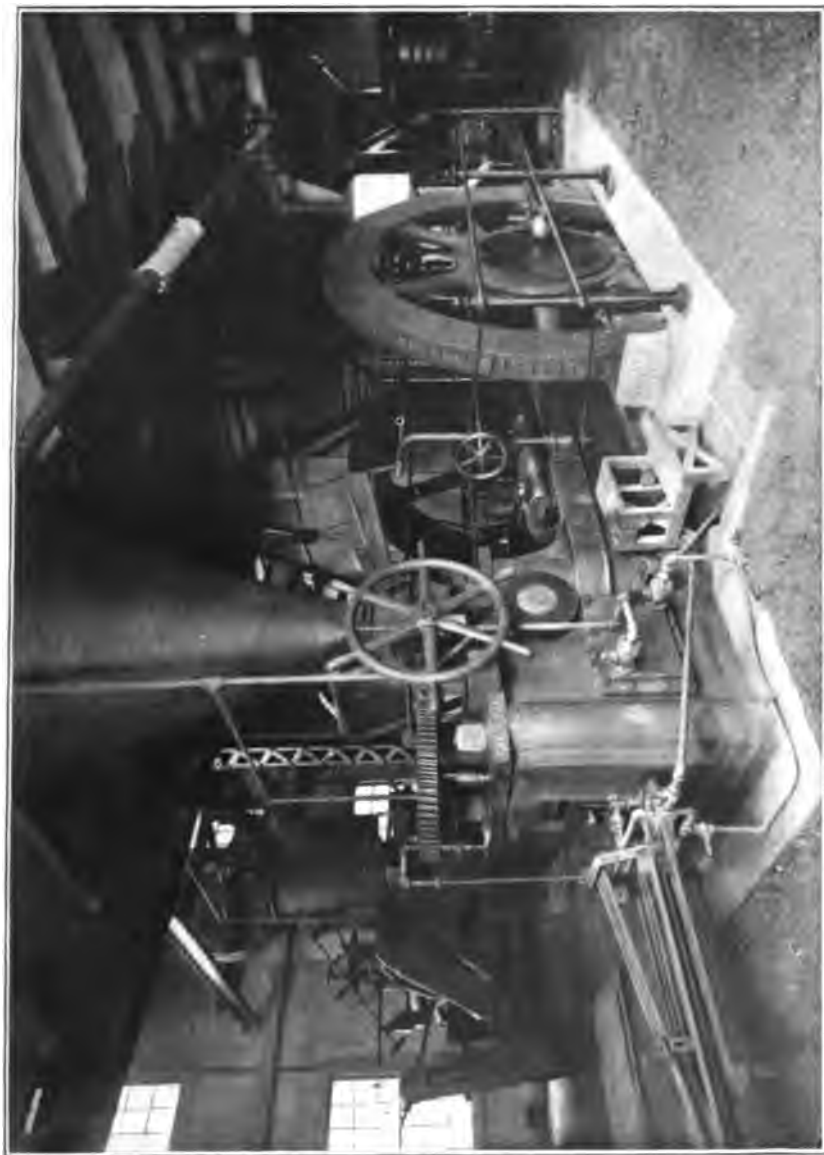
LABORATORY HAND PRESS.

A laboratory hydraulic press (Pl. XII, B), operated by hand, is used for preliminary investigations. It contains a mold 3 inches in diameter and 9 inches long, around which is a steam jacket. This press is adapted for tests of briquets with or without binding material and with hot or cold dies. The pressure obtainable with the 3-inch mold is 50 tons, or 14,000 pounds per square inch. By using a smaller mold (2-inch) a pressure of 32,000 pounds per square inch could be obtained. The press has proved very useful and many tests were made with it, but as the tests with the press were preliminary a discussion of only a part of them is included in this report.

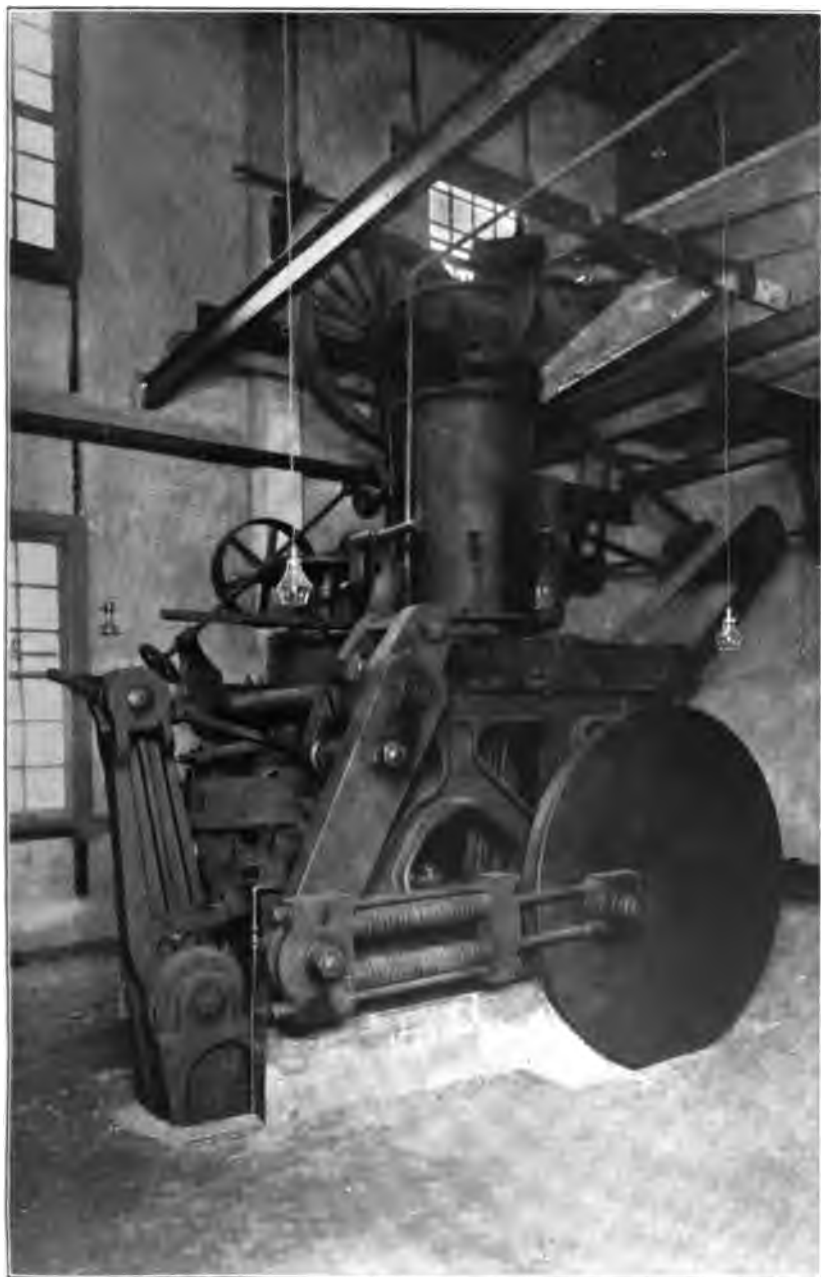
METHOD OF MAKING TESTS ON GERMAN PRESS.

In making a briquetting test about 3 tons of lignite was crushed, dried, and screened by the apparatus previously described for each machine.

In each test the raw material as it was fed into the drier was sampled for chemical analysis and determination of heating value. The dried and cooled material was sampled as it fell into the hopper (K, fig. 2) and its temperature and moisture content were determined. A sample of the briquets was taken for analysis and determination



GERMAN BRIQUETTING MACHINE; ENGLISH MACHINE IN BACKGROUND, AT LEFT.



SIDE VIEW OF ENGLISH MACHINE AT PITTSBURGH.

of heating value. Sizing tests of the raw fuel and the mixture were also made from time to time.

The tests on the German machine described in this bulletin were made without the use of any artificial binder. Those lignites that are not made into good briquets in the German press may, perhaps, be made into excellent briquets in a machine adapted to working with an artificial binder. There is opportunity for further experiments in this direction.

Only small lots of lignite could be tested at a time, because the size of the hopper over the press controlled the quantity of material dried. Only enough material was fed to the drier to fill the hopper. The necessity of drying small lots was unfortunate, as about 6 tons of material could be put through in the time needed to get the apparatus warmed to working temperature, and the use of only about 3 tons of lignite for a test resulted in the first part of each lot being dried more than the last part.

DIE ANGLES USED.

The "die angle," or "angle of dies," is the angle to which one end of the long die blocks is ground to decrease the sectional area of the mold, and thus compress the material passing through. This angle may be seen by referring to figure 3, in which the die angle extends from a to b ; the tangent of the angle may be found with sufficient accuracy by dividing the decrease in vertical height of the mold (a minus b) by the length of the part ground off. As the angular measure in degrees does not convey such exact information as do the dimensions of the part removed by grinding, the latter are styled the "die angle" in the notes on the tests. In the tests here reported various die angles were used, ranging from $\frac{1}{80}$ to $\frac{2}{88}$, as shown in the following table:

Dimensions of die angles used.

Decrease in vertical height.	Length of die angle.	Ratio or tangent.	Angle of which ratio is tangent.	Decrease in vertical height.	Length of die angle.	Ratio or tangent.	Angle of which ratio is tangent.
<i>Mm.</i>	<i>Mm.</i>		<i>° ' "</i>	<i>Mm.</i>	<i>Mm.</i>		<i>° ' "</i>
1.0	60	0.016	0 55	5.0	89	0.0562	3 13
1.5	60	.025	1 26	5.5	89	.0618	3 32
3.0	76	.0394	2 15	6.0	89	.0674	3 51
4.0	110	.0364	2 5	7.0	89	.0786	4 30
4.0	89	.0450	2 35	8.0	68	.1176	6 42

PRESSURES USED.

The German briquetting press was built to give a working pressure of 14,000 to 28,000 pounds per square inch, and as the area of the face of a briquet was 15.55 square inches, the total pressure developed by the machine was 217,700 to 435,400 pounds. The pressure actually used in the different tests could not be determined, and could be approximated only when the material stalled the machine, the pressure then being maximum.

LADLEY BRIQUETTING PRESS.

Through the courtesy of the Indianapolis Pressed Fuel Co. the use of their briquetting plant was obtained to carry out some tests of California lignite without binder and of Philippine lignite with binder at a time when the plant of the bureau was not in condition to make tests. It was also desirable to determine whether an American-made machine would give as satisfactory results as foreign-made machines for briquetting American fuel. The Ladley press at the above company's plant is of the rotary-plunger type and has two rows of molds, 54 molds in each row, arranged in the heavy rim of a wheel. The machine exerts a pressure of 5,000 to 6,000 pounds per square inch. The construction of the press is shown by the views in Plate XV.

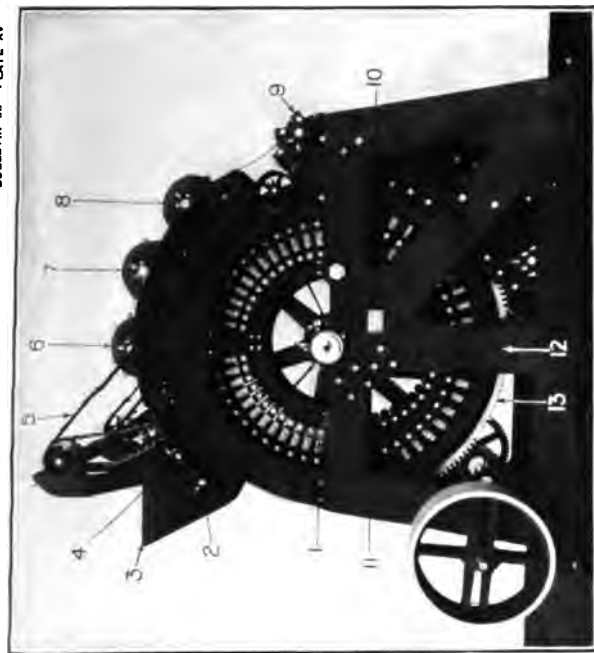
FUELS TESTED.

At the three experiment stations various fuels from different sections of the United States have been subjected to briquetting tests. The following table indicates the kind of fuels tested, the localities from which the shipments were received, the name of the experiment station at which they were tested for briquetting qualities, and the pages of this bulletin on which a description of the tests made are reported. A few of the coals were used only in tests of binders, as reported in Bulletin 24.^a

^a Mills, J. E., Binders for coal briquets; investigations made at the fuel-testing plant, St. Louis, Mo. Reprint of U. S. Geol. Survey Bull. 343.



4. FRONT VIEW OF LADLEY BRIQUETTING PRESS.



B. SIDE VIEW.

1, One of the 108 rams; 2 and 4, shafts of mold-filling device; 3, feed box; 5, belt conveyor to return superfluous dust to feed box; 6, 7, 8, tamping wheels to fill molds under pressure; 9, link or anvil-blocks belt closing outer end of molds to permit application of pressure by rams; travels tangentially to mold wheel during part of revolution; 10, cams moving rams are behind frame at this point; 11, relief springs holding cars in place are attached to frame at this point; 12, point at which briquets are ejected from molds; 13, gear turning mold wheel.

Summary of fuels tested.

State and field No.	Kind and size of fuel.	Name of bed.	Locality.	County.	Railroad.	Station at which briquetting tests were made.	Page of bulletin on which tests are reported.
Alabama:							
No. 1.....	Bituminous, lump and nut.....	Mary Lee or Horse Creek.	1½ miles west of Horse Creek.	Walker.....	Frisco system.....	St. Louis.....	121
No. 2-B.....	Bituminous, run of mine.....	Jagger.....	Carbon Hill.	do.....	do.....	do.....	122
No. 3.....	do.....	Youngblood or Coke.	Belle Ellen.	Fibb.....	Louisville & Nashville.	do.....	123
Argentina, No. 1.....	do.....			Province of Mendoza.	do.....	do.....	125
Arizona:							
No. 1.....	Semibituminous, over 1-inch screen.....	Huntington.	Huntington.	Sebastian.....	Frisco system.....	do.....	126
No. 2.....	do.....	Hartshorne.	Bona.	do.....	do.....	do.....	126
No. 3.....	do.....	Jennie Lind.	Denning.	do.....	Missouri Pacific.	do.....	127
No. 4.....	Semibituminous, slack.....	Denning.	West of Coal Hill.	Franklin.....	do.....	do.....	128
No. 5.....	Semibituminous, half slack, half lump.....	do.....		Johnson.....	do.....	do.....	130
No. 6.....	Bituminous, slack coal.....	Jennie Lind.	Jennie Lind.	Sebastian.....	do.....	do.....	132
No. 7-A.....	Semibituminous, lump, over 2-inch perforated screen.....	Hartshorne.	Midland (4 miles south-west).	do.....	Midland Valley.	do.....	(a)
No. 7-B.....	Semibituminous, slack, through 2-inch perforated screen.....	do.....	do.....	do.....	do.....	do.....	132
No. 13.....	Bituminous, slack.....	(?).....	Denning.	Franklin.....	St. Louis, Iron Mountain & Southern.	do.....	133
California:							
No. 1.....	Subbituminous, run of mine.....	(?).....	Tesla.....	Alameda.....	Alameda & San Joaquin.	do.....	(a)
Pittsburgh, No. 14.....	Brown lignite, run of mine.....		Ione.....	Amador.....	Southern Pacific.	Pittsburgh.	135
Colorado:							
No. 1.....	Subbituminous, run of mine.....	(?).....	Lafayette.	Boulder.....	Colorado & Southern, Burlington system.	St. Louis.....	143
Illinois:							
No. 1.....	Bituminous, over 1-inch screen.....	Herrin (Belleville, No. 6).	O'Fallen.	St. Clair.....	Belt Terminal of East St. Louis.	do.....	143
No. 4.....	Bituminous, over 2-inch screen.....	do.....	About 1 mile west of Troy.	Madison.....	St. Louis, Troy & Eastern.	do.....	144
No. 5.....	Bituminous, slack.....	(?).....	Collinsville.	do.....	do.....	do.....	(a)
No. 6.....	Bituminous, run of mine.....	Jennie Lind.	Coftown.	Montgomery.....	Toledo, St. Louis & Western.	do.....	(a)
No. 7-C.....	Bituminous, slack.....	Herrin (Belleville, No. 6).	Near Collinsville.	Madison.....	Vandalia.	do.....	(a)
No. 7-D.....	Bituminous, run of mine.....	do.....	do.....	do.....	do.....	do.....	(a)
No. 7-E.....	Bituminous, screened nut.....	do.....	Collinsville.	do.....	do.....	do.....	146
No. 8.....	Bituminous, nut.....	(?).....	Falsley.	Montgomery.....	Big Four.	do.....	(a)

a Tests described in Bull. 24, Bureau of Mines.

Summary of fuels tested—Continued..

State and field No.	Kind and size of fuel.	Name of bed.	Locality.	County.	Railroad.	Station at which briquetting tests were made.	Page of bulletin on which tests were reported.
Illinois—Continued.							
No. 9-A	Bituminous, run of mine.	Herrin (Belleville No. 6).	Near Staunton.	Macoupin.	Litchfield & Madison.	St. Louis.	(a)
No. 9-C	Bituminous, slack.	Big Muddy.	do.	do.	do.	do.	147
No. 10	do.		West Frankfort.	Franklin.	Chicago & Eastern Illinois.	do.	(a)
No. 11-B	Bituminous, one-half run of mine, one-half lump coal.	do.	Near Carversville.	Williamson.	Illinois Central.	do.	(a)
No. 11-C	Bituminous, one-half No. 4 washed coal, one-half No. 5 washed coal.	do.	do.	do.	do.	do.	(a)
No. 12-B	Bituminous, size 5.	do.	Bush.	do.	St. Louis Iron Mountain & Southern.	do.	148
No. 14	Bituminous, lump, over 1½-inch bar screens.	Springfield (No. 5).	East side of Springfield.	Sangamon.	Illinois Central.	do.	(a)
No. 20	Bituminous, screenings.	No. 6.	Staunton.	Macoupin.	Litchfield & Madison.	do.	150
No. 21	Bituminous, lump, over 2½-inch screen.	Herrin (Belleville No. 6).	Troy.	Madison.	St. Louis, Troy & Eastern.	do.	150
No. 23-B	Bituminous, slack, through 2-inch perforated screen.	do.	Donkville.	do.	do.	do.	151
No. 28-A	Bituminous, size 5, washed.	Herrin (No. 6).	Herrin.	Williamson.	Illinois Central.	do.	152
No. 28-B	Bituminous, screenings, raw.	do.	do.	do.	do.	do.	153
No. 29-A	Bituminous, screenings, through 2-inch shaking screen.	Herrin (Belleville No. 6).	Livingston.	Madison.	do.	do.	155
No. 29-B	Bituminous, run of mine.	do.	do.	do.	do.	do.	155
No. 30	Bituminous, nut, through 2-inch and over 2-inch shaking screen.	do.	Shiloh.	St. Clair.	Southern.	do.	156
No. 31	Bituminous, screenings, over 1½-inch shaking screen.	do.	Worden.	do.	Wabash.	do.	158
No. 33	Bituminous, screenings, through 1½-inch shaking screen.	do.	Trenton.	do.	Baltimore & Ohio.	do.	159
Indiana:							
No. 1.	Bituminous, run of mine.	do.	Milled.	Clinton.	Evansville & Terre Haute.	do.	160
No. 1-B.	Bituminous, screenings, through 1-inch screen.	do.	do.	do.	do.	do.	161
No. 2.	Bituminous, run of mine.	No. 5.	Booneville.	Warrick.	Southern.	do.	162
No. 5-B.	Bituminous, screenings, through 1½-inch bar screen.	do.	Hymers.	Sullivan.	Evansville & Terre Haute.	do.	162
No. 6-B.	do.	do.	do.	do.	do.	do.	163
No. 19.	Bituminous, screenings, through 1½-inch stationary bar screen.	Brasil Block (top).	Diamond.	Parke.	Chicago & Eastern Illinois.	do.	165
No. 20.	do.	Brasil Block (bottom).	Brasil.	Clay.	do.	do.	166

[illegible]

(a) Tests described in Bull. 24, Bureau of Mines.

Summary of fuels tested—Continued.

State and field No.	Kind and size of fuel.	Name of bed.	Locality.	County.	Railroad.	Station at which briquetting tests were made.	Page of bulletin on which tests are reported.
Pennsylvania:							
No. 3.....	Anthracite; culm.	Lower Kittanning, B. or Miller.	Wehrum	Indiana.	Pennsylvania.	St. Louis.	195
No. 16.....	Semibituminous; run of mine.	Lower Freeport or D. Miller.	Hastings	Cambria	do.	do.	197
No. 16.....	do.	Pittsburgh	Lloydell	do.	do.	do.	198
No. 18.....	do.	Pittsburgh	Hermine	Westmoreland	do.	do.	199
No. 19.....	Semibituminous.	Lower Kittanning or B.	Seward	do.	do.	do.	203
No. 20.....	Bituminous.	Pittsburgh	Huff	do.	do.	do.	205
No. 22.....	Anthracite; buckwheat size.	Pittsburgh		do.	do.	do.	206
Jamesstown No. 18.....	Anthracite; buckwheat size.	Pittsburgh		do.	do.	Norfolk.	207
Philippine Islands:							
Pittsburgh No. 112.....	Lignite; run of mine.		Batan Island	Philippine Islands.		Pittsburgh.	208
Rhode Island:							
No. 1.....	Anthracite graphite coal; run of mine.	Cranston (?)	Cranston	Providence.	New York, New Haven & Hartford.	St. Louis.	219
Pittsburgh No. 19.....	Graphitic anthracite; lump	Cranston - Narragansett Basin.	do.	do.	do.	Pittsburgh.	219
Tennessee:							
No. 1.....	Bituminous; run of mine.	Mingo or Ralston.	Fork Ridge	Clatsome.	Louisville & Nashville.	St. Louis.	221
No. 4.....	do.	Windrock or Dean.	Oliver Springs	Anderson.	do.	do.	222
No. 7-B.....	Bituminous; slack, through ½ by 1-inch screen.	Wildor.	Wildor	Fentress.	do.	do.	223
No. 9-B.....	Bituminous; raw slack.	Sewanee.	Coalmont.	Grundy.	Nashville, Chattanooga & St. Louis.	do.	225
No. 9-C.....	Bituminous; washed slack.	do.	do.	do.	do.	do.	225
No. 10.....	Bituminous; slack.	Battle Creek.	Orme.	Marion.	do.	do.	226
Texas:							
No. 1.....	Brown lignite; run of mine.	(?)	Wooters Station.	Houston.	International & Great Northern, Kansas & Missouri, Texas.	do.	60
No. 4.....	Lignite; run of mine.	(?)	Hoyt.	Wood.	do.	do.	227
Pittsburgh No. 7.....	Brown lignite; ½-inch lump.	"Big Vein"	3½ miles northeast of Rockdale.	Millam.	International & Great Northern.	Pittsburgh.	228
Pittsburgh No. 8.....	Brown lignite; ½-inch lump.	(?)	2 miles southwest of Lytle.	Medina.	do.	do.	229
Pittsburgh No. 9.....	Brown lignite; ½-inch lump.	Upper.	6 miles west of Calvert.	Robertson.	do.	do.	232
Utah:							
No. 1.....	Bituminous; run of mine.	(?)	Huntingdon Creek.	Carbon.	do.	St. Louis.	235
No. 2.....	Subbituminous; slack, through 11-inch screen.	Coalville, Ten Foot, or Grass Creek.	Coalville.	Summit.	Union Pacific.	do.	236
Pittsburgh No. 36.....	Subbituminous coal.	Washach (?)	do.	do.	do.	Pittsburgh.	91

FUELS TESTED.

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Virginia: No. 5-D.....	Semla anthracite; slack	Rig seam	Blacksburg	Montgomery	Norfolk & Western	St. Louis	238
James town No. 2.....	Semibituminous; run of mine	Pocahontas No. 3.....	Pocahontas	Tazewell	do.	Norfolk	239
James town No. 14.....	Semianthracite; culm	(?).....	Merrimac	Montgomery	do.	do.	241
Washington: No. 2.....	Bituminous; lump	Rodlyn.....	Rodlyn	Kittitas	Northern Pacific	St. Louis	242
Pittsburgh No. 35.....	Subbituminous	Upper Freeport.....	Tono	Thurston	do.	Pittsburgh	93
West Virginia: No. 3.....	Bituminous; run of mine	Fire Creek (Quinnimont)	4 miles southeast of Morgantown	Monongalia	Morgantown & Kingwood	St. Louis	243
No. 6.....	Semibituminous	Sewall	Rush Run	Fayette	Chesapeake & Ohio	do.	243
No. 7.....	do.	Powellton	Sun	do.	do.	do.	(a)
No. 9.....	Bituminous	Thin vein Pocahontas	3 miles south of Powellton	do.	do.	do.	(e)
James town No. 3.....	Semibituminous; run of mine	Sewell No. 5.....	Davy	Hampshire	Norfolk & Western	Norfolk	244
James town No. 6.....	do.	Sewell	1 mile east of Sewell	Fayette	Chesapeake & Ohio	do.	245
James town No. 7.....	do.	do.	Redstar	do.	do.	do.	247
James town No. 8.....	do.	Quinnimont (Fire Creek)	Derryhale	do.	do.	do.	248
James town No. 9.....	do.	Sewell	Lawton	do.	do.	do.	249
James town No. 10.....	do.	Beckley	Winona	do.	do.	do.	251
James town No. 11.....	do.	do.	Stanford	Raleigh	do.	do.	252
James town No. 12.....	do.	do.	West Raleigh	do.	do.	do.	253
James town No. 13.....	do.	Pocahontas No. 3.....	Switchback	McDowell	Norfolk & Western	do.	255
James town No. 15.....	do.	do.	Ennis	do.	do.	do.	256
James town No. 16.....	do.	Fire Creek and Sewell	Minden	Fayette	Chesapeake & Ohio	do.	257
James town No. 17.....	do.	Pocahontas No. 3.....	Midvale	Doddridge	do.	do.	258
Wyoming: No. 1.....	Black lignite; over 8-inch screen	do.	9 miles northwest of Sheridan	Norfolk & Western	do.	do.	260
No. 6.....	Subbituminous; run of mine	(?).....	Sheridan	Burlington system	St. Louis	do.	261
Miscellaneous: No. 5.....	Coke breeze	do.	Kennermer	Union Pacific	do.	do.	263
No. 9.....	do.	do.	Uinta	do.	do.	do.	264
							264

a Tests described in Bull. 24, Bureau of Mines.

BINDERS USED.

In the course of the fuel-briquetting investigations of the fuel-testing plants, attention was of necessity given to binding materials. With very few exceptions the binder tests were confined to primary binding materials, and practically no attention was given to trials of patented binders. Coal-tar pitch has been and is at present the binder most generally used in this country and in foreign countries.

In the briquetting experiments at the three fuel-testing plants a binder called water-gas pitch has been used to a larger extent than any other, as it was available at satisfactory prices and in sufficient quantities. This binder has an advantage over coal-tar pitch in the proportion of free carbon present, which is usually less than 10 per cent, whereas in coal-tar pitch it often amounts to 30 to 40 per cent. As this free carbon or carbon dust has no binding qualities itself, it acts merely as a diluent of the pitch, and therefore the binder that possesses the lower proportion of free carbon should have the greater binding power, other factors being equal. On the other hand, good grades of coal-tar pitch may possess a higher percentage of the heavier distillates than water-gas pitches and thus offset the disadvantage of a high percentage of free carbon.

In general, it may be said that the best grades of coal-tar pitch for fuel-briquetting purposes are those made by distilling the pitch to such a point that some of the anthracene or "green oils" remain with the hard pitch in the still rather than by distilling all the oil and then reducing the hard pitch with the naphthalene or creosote oils, as is sometimes done in preparing "reduced" pitches.

The quality or hardness of the pitch to be used will depend on the location of the plant and the time of year at which the pitch is to be used. Generally speaking, that grade of pitch known as medium hard pitch is most desirable for briquetting fuels. For use in summer or in the hotter parts of the country in the winter months, a harder pitch should be used than in the winter or in the colder parts of the country.

There are two general methods of incorporating pitch binder with fuel to prepare a mixture for briquetting. The most general method is the one in which the pitch is ground and mixed with the fuel and the mixture heated by steam or air until the pitch is softened sufficiently to flow. In the other method, the pitch is melted separately from the fuel and the latter is heated to a temperature above the melting point of the binder, so that when they are mixed the pitch is easily and evenly distributed throughout the mass.

Although authorities differ as to the best manner of mixing pitch binder with the fuel, the bureau's experiments indicate that less pitch is required if it is not melted but only softened sufficiently to be "tacky," and the particles of coal are made to adhere by the particles of pitch

and the pressure of the machine. An advantage derived from not heating the mixture too hot is that the briquet, when leaving the machine, will be cold enough so that the pitch will be somewhat hard, thus giving a maximum strength. Otherwise when the pressure is relieved there is a tendency for the compressed air and steam in the briquet to expand and cause cracks on its surface. There is a temperature for each grade of pitch that will give the greatest plasticity to the mixture and at the same time the greatest strength to the fresh briquet. With some types of equipment the mixture of fuel and pitch passes through a spiral cooling conveyor after leaving the heater and before reaching the press, and with this arrangement it is possible to "temper" the mixture to the desired consistency.

If pitch be melted to a thin liquid some of it will be absorbed by the coal particles, and although this result might be desirable when briquetting a fuel like lignite, it is not necessary and is therefore a waste of material in the case of anthracite and bituminous coals. The use of more pitch binder than is absolutely necessary to stick the particles together increases the cost of briquetting and adds to the smoke-producing qualities of the briquets.

Wherever possible it is best to mix the larger pieces of pitch and coal and pass them through the fine grinder together. By this treatment a more intimate mixture is obtained than if each is ground separately and mixed afterwards. It is much easier (especially in warm weather) to grind the pitch in mixture with the coal than by itself.

The tests applied to binding materials changed as the work progressed, so that the earlier tests were not made along the same lines as the later ones. Therefore the results of all of the tests of binders can not be included in one table. A description^a of the binder tests made in the first part of the work at St. Louis is given here and the tabulated data of the later tests are given elsewhere.

BINDERS USED IN EARLIER TESTS AT ST. LOUIS.

It was found that one binder would give good results with one coal but could not be used with other coals. On account of the situation of the coal districts it is often necessary, for commercial reasons, to substitute another binder for one that under laboratory conditions would give good results. An attempt was made not only to obtain a binder with which the coal could be briquetted, but to devise a binder that would make briquetting commercially possible. The greatest difficulty was experienced with the various grades of lignite, most deposits of which are too remote from supplies of available binder to make it possible to utilize certain materials for binding purposes that would otherwise give satisfactory results.

^a Taken from Pratt, J. H., Briquetting tests: U. S. Geol. Survey Prof. Paper 48 (pt. 3), 1906, pp. 1394-1401.

The materials that have been tried as binders in the laboratory experiments and in the two briquetting machines include the following: Pitch of various grades, creosote, asphalt (hard and soft, crude and refined), asphaltic pitch, petroleum (both of paraffin and asphalt bases), molasses, lime, and clay.

Pitch is the residue left from the distillation of tar that is obtained (1) from by-product coke ovens, (2) from illuminating-gas plants, (3) from producer-gas plants, (4) from Pintsch-gas plants, (5) from water-gas plants, and (6) from petroleum-gas plants. Pitches made from the first three varieties of tar were tested and gave equally good satisfaction when approximately of the same composition. Petroleum-gas tar gave the best results.

There is considerable variation in the tars obtained by the various processes mentioned above, the pitch obtained from different batches of the same tar often varying greatly even when distilled to the same final temperature. In order to obtain a uniform grade of pitch it is necessary to keep the fire at approximately the same intensity, and it is usually considered advantageous to use the same still for the same tar. Tar obtained from by-product coke ovens differs in quality from tar made at illuminating-gas works, probably because the condensers of the former are kept at a temperature so high that the available benzole does not enter the tar, but is removed separately. It usually contains much more fixed carbon than that obtained from gas-house tars. It also contains more water than the other tars.

Pintsch-gas tar is obtained as a thin emulsion in water, from which it can be separated only by long standing in a tank at a temperature just below 212° F. The tar is composed largely of naphthalene and benzine.

The entire output of tar from many illuminating-gas and by-product coke plants is now purchased by pitch manufacturers. The price of coal-gas tar varies considerably, reaching at times 3 cents per gallon. The price of illuminating-gas tar, guaranteed to contain less than 10 per cent moisture, is 6 cents per gallon. An average sample of this tar contains 5.25 per cent of water, gives 14,172 British thermal units, and has a specific gravity of 1.21. Water-gas tar is sold at 4 cents per gallon, averages 6.75 per cent of water, and gives 15,325 British thermal units. Its specific gravity is 1.12. At present there is no market for Pintsch-gas tar, which has always been sold cheaper than water-gas tar. When dried, this tar contains 4 per cent of water, gives 12,304 British thermal units, and has a specific gravity of 1.18.

The pitches submitted to the plant for the experimental work varied considerably. The first pitch received was from the St. Louis works of the Barrett Manufacturing Co. The pitch was too

hard and would not soften at a temperature low enough to be used satisfactorily in either the English or the American machine. The pitch did not contain a sufficient proportion of the creosote oils to give it the required binding qualities. Throughout this report this pitch is designated as pitch A. The second lot of pitch was received from the Chatfield Manufacturing Co., of Carthage, Ohio, and was somewhat harder than pitch A, indicating that a still greater percentage of the creosote and lighter oils had been driven off. This pitch is designated throughout this report as pitch X. Although these pitches would make briquets, it was necessary to use 13 to 18 per cent of them, whereas only 6 to 9 per cent of pitch containing the proper amount of light oils would be required with the same coals.

In order to obtain a pitch that would be satisfactory, some of the best tar from the producer-gas plant was heated until all the water was expelled. The boiling was continued until the residue became brittle when dropped into water having a temperature of 55° F. After bucking down, this pitch stuck together on standing one week in the laboratory. It is designated throughout this report as pitch Z. A long series of tests was then made with this pitch and various coals, and 6 per cent of it was found to make better briquets than 13 per cent of pitch X.

In order to demonstrate further that the right quality of pitch required a higher percentage of the creosote or adhesive oils than pitches A or X, experiments were made with hard pitch A and water-free producer-gas tar. These were combined in various proportions until a product was obtained that became brittle when dropped into water at 55° F. A sufficient quantity of this product to briquet 2 tons of coal was made, and it was found to give the results desired. This pitch will soften when held in the mouth; when first bitten it cracks and crumbles like spruce gum, but almost immediately becomes plastic and can be chewed like ordinary gum. This simple test, in nearly all cases, will determine whether a pitch is of the proper quality for use in briquetting. This pitch is designated throughout this report as pitch Y. Samples were sent to the Barrett Manufacturing Co. and the Chatfield Manufacturing Co. for duplication in quantity. The pitches received later from the Barrett Manufacturing Co. are designated in this report as pitches B, C, and D. Those received from the Chatfield Manufacturing Co. are known as pitches E, F, and G.

Another pitch that was furnished by the Barrett Manufacturing Co. was obtained as a by-product from the manufacture of gas from heavy petroleum. When hot this pitch has a marked odor of kerosene. It is lustrous and works well on a cool day, but is a little too soft in ordinary weather for experiments on the English machine.

This pitch is designated as pitch H throughout this report. Analyses of these pitches were made by E. E. Somermeier, with the following results:

Proximate analyses of pitches.

Kind of pitch.	Laboratory No.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.
		<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
A	1161	0.47	47.93	50.79	0.81	^a 0.62
B	1311	1.14	49.66	47.88	1.32	.70
C	1391	1.88	62.75	35.84	.53	.57
D	1466	1.45	54.05	43.91	.59	^a .56
E	1464	1.02	54.11	44.04	.83	^a .66
F	1457	.57	52.98	45.31	1.14	^a .80
G	1458	.60	52.53	45.62	1.25	^a .70
H	1555	1.04	61.44	36.72	.80	-----
X	1125	.33	59.07	39.44	1.16	.88

^a Approximately determined.

Ultimate analyses of pitches.

Kind of pitch.	Laboratory No.	Hydrogen.	Carbon.	Nitrogen.	Oxygen.	Sulphur.	Calorific value.
		<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	
A	1161	4.17	91.30	0.85	2.25	^a 0.62	-----
B	1311	3.97	90.89	1.05	2.07	-----	8,782
C	1391	4.72	91.16	1.16	1.85	.58	8,937
D	1466	4.28	91.57	1.10	1.90	.56	-----
E	1464	4.22	91.30	1.00	1.99	.66	-----
F	1457	4.16	91.50	1.00	1.40	.80	-----
G	1453	4.06	90.82	1.01	2.16	.70	-----
X	1125	4.56	90.34	.99	2.07	.88	8,820

^a Determined by Eschka method.

Although no definite conclusions can be drawn from these analyses, yet of the pitches distilled from the same tar, under like conditions, those with the highest percentage of volatile matter gave the best results in briquetting. Pitches E, F, and G, furnished by the Chatfield Manufacturing Co., made from the same tar and distilled under approximately the same conditions, showed binding power that increased from G to E according to the increase in the percentage of volatile matter and the decrease in the percentage of fixed carbon. Pitch B, which was made under entirely different conditions and from a different tar, is much lower in volatile matter and higher in fixed carbon than E, yet makes a better pitch for briquetting than either E, F, or G.

In order to obtain a more definite estimate of the influence of the composition of pitches on their binding qualities and their value in briquetting, a series of tests was made by distilling their volatile contents. The results are given in the following table:

Distillation of pitches.

Kind of pitch.	Laboratory No.	Quantity of pitch used.	Temperature at which first drop came over.	Temperature at which the last of the oil was observed to come off.	Percentage of oil.	Final temperature to which still was raised.	Residue.
A	1161	<i>Grams.</i> 200	<i>° F.</i> 380	<i>° F.</i> 385	1.38	<i>° F.</i> 400	Hard coke.
B	1311	200	351	...	7.38	400	Do.
D	1465	200	332	375	13.01	400	Do.
E	1464	200	350	370	1.61	400	Do.
G	1453	200	324	382	.96	400	Do.
X	1125	200	391	...	.71	400	Do.

The total percentage of oils given off during the distillation was determined with sufficient accuracy for comparative purposes. It will be seen from the above table that there is a decided increase in the percentage of oils in the pitches that gave the best results. Pitch X, which was the hardest pitch tested and gave the least satisfaction, contains only 0.71 per cent of volatile oil, whereas pitch D, which was the most satisfactory of those given above, contained 13.01 per cent of oils. Pitches A and E, made by different companies but very much alike and of about the same grade, were not good pitches for briquetting. Pitch G was harder than either A or E, and little, if any, better than pitch X. Pitch B, which was one of the better ones used, approaching D in quality, contained 7.38 per cent of volatile oils. The experiments have shown that from 7½ to 14 per cent of these oils is necessary in the pitches in order to give them the binding qualities desired in briquetting. Pitches with a higher percentage of these oils are too soft and can not be used to advantage. Again, the oils driven off at 315° F. and lower temperatures are the creosote oils, for which there is a large demand. They command a price of 5½ to 6 cents per gallon, or about \$15 per ton. Creosote was mixed with some of the hard pitches for binding purposes, and the product gave satisfactory results.

ROSIN.

The cheaper grades of rosin can be used for binding purposes to as good advantage as the refined material. Rosin may, in some cases, be used to advantage for hardening as well as for binding purposes. It has been used with lime and pitch, and good briquets were made with this combination. It was found that a better quality of briquets could be made with a smaller percentage of a mixture of rosin and pitch as a binder than with pitch alone. In burning, there is little or no odor from the rosin, although there is some tendency to smoke. Lime was used with the rosin to prevent, if possible, the smoking of the rosin briquets.^a

^a The present price of rosin, which approximates \$30 a ton, indicates that it can not compete with pitch or binding purposes.

ASPHALT.

Refined California asphaltum of grade B was used in the experimental work, and is designated throughout this report as asphalt B1. Another asphalt product used had a rubbery consistency and is known as "kopak" No. 30. It is manufactured by the Raven Mining Co., of Texas, and is designated in this report as asphalt B2. An asphaltic pitch made by the Waters-Pierce Oil Co. at one of its Texas plants was also used and is designated as asphalt B3. Crude asphalt from Indian Territory (Oklahoma) is designated B4. The last-mentioned asphalt is altogether too hard, approaching a coke and containing only a small percentage of volatile matter. A soft asphalt was received from the Gulf Refining Co., of Port Arthur, Tex., and is designated throughout this report as asphalt B5. An analysis of this asphalt by E. E. Somermeier was as follows:

Analysis of soft asphalt (B5) from the Gulf Refining Co.

	Per cent.
Moisture.....	0.00
Volatile matter.....	80.75
Fixed carbon.....	19.25
Ash.....	.00
Total.....	100.00

Another soft asphalt was furnished by John McNeil, Casper, Wyo., and is known throughout this report as asphalt B6. It was a soft, tough asphalt, which on cooling to about 40° F. became brittle. An analysis by E. E. Somermeier gave the following results:

Analysis of asphalt (B6) from Casper, Wyo.

	Per cent.
Moisture.....	0.48
Volatile matter.....	78.77
Fixed carbon.....	20.75
Ash.....	.00
Total.....	100.00

PETROLEUM.

Petroleums of both paraffin and asphaltum bases may sometimes be used to advantage in briquetting. On account of lack of time, little experimental work was done with the petroleums. Those that were used were crude oil from Spindletop, Tex., produced by the Gulf Refining Co., and a petroleum, also from the Beaumont district, produced by the Great Southern Refining Co. The latter had a gravity of 15° B., with a flash test of 505 and fire test of 570. The two petroleums mentioned are designated P1 and P2 throughout this report. The only other oil that was tested in briquetting was a Kansas crude oil, which is designated throughout this report as P3.

MOLASSES.

A few experiments were made with molasses in order to determine the possibility of utilizing waste products from cane and beet sugar refineries in briquetting lignite and lignite coals that occur in the vicinity of the refineries. Although no positive results have been obtained with such waste products as a binder, the work has shown the probability that they can be so utilized.

LIME.

Lime was used in combination with both rosin and molasses and some encouraging results were obtained. With rosin the lime was used principally to prevent smoking. One of the main objections to using more than a small proportion of lime is the fact that it increases the percentage of ash in the fuel without raising its heating efficiency. On the other hand, it is one of the cheapest binders and may be obtained at places easily accessible to most of the coal districts.

CLAY.

A few experiments were made with clay in combination with other compounds in an attempt to obtain a suitable binder for lignite coals, but without success.

LABORATORY EXPERIMENTS AT ST. LOUIS.

In testing the value of the various binders for the briquetting of coal, a series of experiments with each was made in the laboratory by briquetting the mixtures on a hand press. Each mixture of coal and binder was heated in a glass assay crucible over a Bunsen burner. In most cases it was heated only to about 212° F., but when the binder was softened with difficulty the temperature was increased. Until the regular hand press was completed, the experiments were made on a screw press with a steel mold about three-eighths of an inch in diameter.

The various coals, after they had been heated with the different binders, were tested in the press, with varying pressures on the same mixtures. Thus, for each mixture there was a series of six briquets made, representing, respectively, a light pressure, a moderate pressure, a pressure of 2,500 pounds, a pressure of 4,000 pounds, and the maximum pressure of about 5,000 pounds. The quantity of mixture fed into the mold was also varied, so that some of the briquets made were only one-fourth of an inch long, whereas others were an inch long. All the briquets made with a particular mixture gave practically identical results with the same length of briquet, and there was usually only slight variation with briquets made under different pressures.

PITCH TESTS.

GENERAL RESULTS.

Special experiments were made with the different pitches, A, B, etc., and if there was considerable variation in their composition more complete experiments were made. All of the pitches mix with gas tars and creosote, making a product that resembles the softer pitches. They also mix with rosin in any proportion, giving a mass having the expected intermediate properties. They will not mix with any of the Kansas or Texas crude oils or the refined Texas petroleum (P2), or with Indian Territory (Oklahoma) asphalt (B4), California asphalt (B1), Texas asphalt (B2), asphaltic pitch (B3), or soft asphalt (B5). Experiments showed that rosin and oil would mix in all proportions, and such mixed masses of rosin and oil were tried in combination with the pitches, but on adding the pitch to this combined oil and rosin the oil was liberated and the resultant mass was granular and greasy, without any strength whatever. Another experiment showed that crude oil would neutralize the effect of clay in a coal, and although it was hoped that a combination of oil and pitch might be used, it was found that the crude-oil products would not mix with any of the tar pitches, and thus no positive results could be obtained by such combination.

The experiments made with pitch as a binder, when heated to 212° F., indicate that the results are better when there is a slight quantity of steam present. When coal and pitch, mixed with a small quantity of water, were heated in a clay crucible until steam began to come off freely and were then pressed, much better results were obtained than when the two were heated dry to the same temperature, or when the wet mixture was heated until all the water had been evaporated.

TESTS OF PITCH A.

Experiments were made in the laboratory with pitch A to determine its binding qualities. Nine per cent of this pitch was mixed with Arkansas No. 1 coal and squeezed to the limit of the screw press. The warm mixture yielded a highly polished cylindrical briquet, which, after remaining in a beaker of boiling water for two hours, showed no noticeable softening. This same mixture was further heated until the pitch began to smoke and was then put through the screw press. The resulting briquet was smooth and clean.

Seven per cent of pitch A with Arkansas No. 4 coal, when heated in an assay crucible and run through the hand press, made a fair briquet which burned well. Six per cent of this same pitch with Arkansas No. 6 coal, heated in the same way and with the maximum pressure of the hand press, made a crumbly briquet. Alabama No. 1

coal was also tested with pitch A, the coal having been previously crushed to pass a 60-mesh sieve. With 7 per cent of pitch A, when the mixture was heated so as nearly to burn the fingers, the resultant briquets were just beginning to cohere. With 9 per cent it made a well-pressed, coherent, but rough briquet. With 11½ per cent the briquets were little or no better, except that they were somewhat cleaner. With 20 per cent, when the mixture was only warm, but with a maximum pressure, the resultant briquet was good. All these briquets were, however, dirty.

The above experiments show that the best practical results with this hard pitch were obtained by using 9 per cent of it with a coal easily briquetted; with other coals the percentage should be much higher.

Pitch A was further tested with rosin, but no satisfactory results were obtained with pitch A and rosin.

TESTS OF PITCH B.

Pitch B was tested with West Virginia coking coal. With 3 per cent of pitch B a fair briquet was obtained. With 4 per cent the resulting briquet was a little better, and with 5 per cent a good briquet was made which had the desired lustrous fracture. With Arkansas No. 4 coal 6 per cent of this pitch was needed to give the desired results, and as this coal represents more nearly the character of the coals to be briquetted, 6 per cent was accepted as the percentage required of this and the similar pitches C and D to make a good briquet. A series of experiments was also made with this pitch and lime. The experiments indicated that there is no advantage in using lime with pitch in an attempt to reduce the percentage of pitch. If, however, some third substance that will react with the lime can be used it may be possible thus to produce a binder that will be cheaper than pitch alone and quite as satisfactory.

TESTS OF PITCHES C AND D.

Pitches C and D, which are similar to pitch B, were used with the formulas derived for B, and in nearly all cases the runs made on the English and American machines were satisfactory.

TESTS OF PITCH E.

Pitch E, which melts at 180° F., is a trifle too hard for satisfactory briquetting on the English machine and is not equal to pitches B, C, or D. Six, 7, 8, and 9 per cent of this pitch were tested alone on Illinois No. 4 coal. With 6 per cent the briquet was noncoherent; with 9 per cent an excellent briquet was made. With Arkansas No. 5 coal, which makes an excellent briquet with 6 per cent of pitch B, 8 per cent of pitch E was necessary to obtain a corresponding briquet.

Another series of experiments with pitch E was made with Kentucky No. 2 coal. The coal was crushed so that half would pass through a 20-mesh sieve and half through a 10-mesh sieve. With 7 per cent of pitch E a briquet was obtained that was just coherent; with 8 per cent the briquet was nearly satisfactory; but with 9 per cent an excellent briquet was obtained, similar to those made with the Illinois No. 4 coal.

TESTS OF PITCH G.

Pitch G, which melts at 196° F., is too hard (as are A and X) to be used alone in briquetting. With 1½ per cent of creosote and 8 per cent of pitch G, a good briquet was obtained that disintegrated only slightly in the fire, and this formula was used in making a trial run on the English machine.

TESTS OF PITCH X.

Experiments were made with pitch X with 70 per cent of Pennsylvania No. 2 anthracite coal and 30 per cent of West Virginia No. 1 bituminous coal. To these mixed coals were added varying percentages of pitch X, until with 13 per cent the briquets were very satisfactory. Briquets were also made by using 12 per cent of pitch X with a mixture of coal containing 80 per cent of the anthracite and 20 per cent of the bituminous, and with a mixture of 60 per cent of the anthracite and 40 per cent of the bituminous. All of these mixtures made briquets that burned without any disintegration in a muffle. Pennsylvania No. 2 anthracite coal was also tested alone with 12 per cent of pitch and the resultant briquet was satisfactory. In all of the above cases the mixture was heated in an assay crucible until the pitch softened. No experiments on the English machine, however, could be made with these combinations.

TESTS OF PITCH Z.

The gas-producer pitch (Z) was tested with Arkansas No. 3 coal by using 6.7 per cent of the pitch. While hot these briquets were soft, but on standing one hour they became very hard and had a glossy or lustrous fracture. When tested in the muffle they burned without disintegration.

TESTS OF ASPHALT BINDERS.

Asphalts were used alone as a binder and also in combination with other compounds. A number of satisfactory results were obtained in the laboratory work, but only a few of the asphalts could be tried on either the English or the American machines, neither of which was adapted to the use of such mixtures.

RESULTS WITH CALIFORNIA ASPHALT.

California asphalt (B1) is brittle at ordinary temperatures and can be readily disintegrated. At 212° F. it softened slightly. It mixed readily with the Spindletop, Tex., petroleum (P1), and the residual product of Beaumont petroleum (P2) in all proportions, and can therefore be thinned to any consistency. A series of experiments was first made with 4 per cent of this asphalt with Arkansas No. 5 coal, from which a fair briquet was obtained. With 5 per cent the briquet was satisfactory, and with 6 per cent it was only a little better. By using 6 per cent of the asphalt, moistening the mixture, and then heating until steam came off, a briquet was obtained that was intermediate in quality between those made with 4 and 5 per cent mixed dry, and raised to a higher temperature. This indicates that to obtain the best results with this asphalt it would be necessary either to heat the mixture with superheated steam or to add crude oil.

The California asphalt was further tested with the Indian Territory No. 6 coal, and with 10 per cent of the asphalt an excellent briquet was obtained that was strong and sufficiently hard, but showed many clayey streaks. The Indian Territory briquets burned fairly well in the muffle.

RESULTS WITH "KOPAK."

The Texas asphaltic compound known as "kopak" (B2) softened slightly at 212° F. without becoming sticky. It melted at a rather high temperature, but was very adhesive. It can not be disintegrated readily in warm weather, but at a low temperature it can, with some care, be crushed. It is not affected by the crude Texas oil (P1) or the residual petroleum from Texas (P2). With the warm asphaltic pitch it mixed readily and little of this pitch was needed to make a soft, very sticky mass. With ordinary coal-tar pitches this asphaltic compound did not mix at all. With rosin the rubbery asphalt mixed readily, and the resulting product could be cracked rather easily. This asphalt has qualities that should make it a desirable binder, and it may perhaps be advantageously used in briquetting lignites.

The B2 asphalt was tested with a dry Texas lignite, 12 per cent of asphalt, heated, being used as a binder. The resulting briquet was tough and strong, but did not burn well, although it had more cohesion than a lignite briquet made with pitch. By using 12 per cent and heating with steam in the usual manner a briquet was obtained that was fully as strong as the one heated to a higher temperature. When the mold was warm a black, glossy briquet was produced. With 10 per cent of asphalt there was seemingly no difference in the appearance of the briquets. They disintegrated in the

furnace, however, and for this reason this line of experimentation was not pursued further.

This asphalt was further tested by using 6 per cent in combination with 4 per cent of the asphaltic pitch (B3) and the dry Texas lignite. The resultant briquet was soft when taken from the mold. After standing one hour it presented a fair appearance, but had no coherence when burning, showing that no advantage is to be gained by adding the asphaltic pitch.

RESULTS WITH ASPHALTIC PITCH.

Asphaltic pitch (B3), made by the Waters-Pierce Oil Co. at one of its Texas plants, has excellent binding qualities and the right consistency to work easily. The pitch was tested first with Arkansas No. 5 coal, by using 6 per cent of pitch as a binder and heating to a moderate temperature in an assay crucible. The briquets were somewhat porous and not strong. By using 8 per cent of the pitch and moderate pressure a good briquet which stood up well in the fire was obtained. With Arkansas No. 6 coal, 6 per cent of this pitch made a tough, elastic, and rather soft briquet. This pitch does not give the degree of hardness that can be obtained by using coal-tar pitches.

Another test was made with Arkansas No. 5 coal by using 2 per cent of the asphaltic pitch and 4 per cent of lime as binder. The briquets showed little or no coherence when taken from the mold and no change in physical character at the end of 40 hours. Three per cent of pitch and 3 per cent of lime made a briquet that was only slightly better. Four per cent of pitch and 4 per cent of lime made a fair briquet, which was improved a little by an increase of pressure. Four per cent of pitch and 3 per cent of lime gave practically the same result. As there was seemingly no reaction between the pitch and the lime no advantage was gained in using lime.

A series of experiments was made with 5 per cent of the pitch and 1 per cent of rosin and Arkansas No. 2 coal. The briquet was somewhat soft when taken from the mold and did not harden in 24 hours. With 4 per cent of pitch and 2 per cent of rosin a briquet was obtained that was satisfactory in every way. Three per cent of pitch and 3 per cent of rosin gave seemingly the same results as 4 per cent of pitch and 2 per cent of rosin. With 2 per cent of pitch and 4 per cent of rosin a good briquet was obtained, which, however, was slightly brittle. One per cent of pitch and 5 per cent of rosin made a hard and strong briquet, but it was more brittle than the preceding. The briquets made from 4 per cent of asphaltic pitch and 2 per cent of rosin, and from 3 per cent of pitch and 3 per cent of rosin were of equal value. The 3-to-3 briquet is probably cheaper, and this ratio was adopted.

RESULTS WITH INDIAN TERRITORY ASPHALT.

Indian Territory asphalt (B4) did not soften in boiling water. It melted to a stiff, sticky liquid just below its ignition point. It is only slightly soluble in gasoline. At the boiling point there was seemingly no reaction between this hard asphalt and crude petroleum P1, P2, and P3. Neither was there any apparent reaction between this asphalt and melted rosin. With creosote it softened materially and when used with this in the ratio of 5 per cent of each with 90 per cent of Arkansas coal a fair briquet was obtained. With Pintsch-gas tar an apparent reaction took place which caused the separation of some of the contained water. One of the first tests made with this asphalt was to determine whether Kansas oil would have any softening effect on the asphalt, but no matter what quantity of oil was used, when heated with steam, only a noncoherent, greasy mass was formed. The asphalt was also tried alone with Arkansas coal by using 4 per cent of asphalt. The mixture was used dry and worked at a high temperature, but the resulting briquet was poorly coherent. With 6 per cent of asphalt the briquet was little or no better.

RESULTS WITH WYOMING ASPHALT.

Wyoming asphalt (B6) united readily with rosin. Two parts of asphalt and one part of rosin were melted together in a kettle and then cooled overnight. On a cool day the mass was very brittle, but became sticky with a rise in temperature. In using the mixture one to two parts of lime were added to prevent its becoming too soft while hot. The briquets crumbled easily in burning: A Wyoming lignite, undried, was tested with 8 per cent of the mixture (2 parts Wyoming asphalt with 1 part of rosin) and 1 per cent of slaked lime. This made a good briquet, which burned fairly well. With 7 per cent of the mixture and 1 per cent of lime the result was practically the same. With 6 per cent of the mixture and 1 per cent of lime the briquet was crumbly. A briquet made with 7 per cent of the mixture alone without lime appeared a little tougher when taken from the mold, but it did not burn so well. Seven per cent of the mixture and 2 per cent of slaked lime gave results similar to those with 7 per cent of mixture and 1 per cent of lime.

In testing this asphalt alone it was necessary to heat the coal in an assay crucible, add the fragments of asphalt, knead them together, and heat frequently. Six per cent of this asphalt alone made with Wyoming lignite a strong, tough briquet which was slightly earthy in appearance. When tested in the fire, however, the briquet fell to pieces at once.

ROSIN TESTS.

Rosin used alone as a binder produces brittle briquets. The results of the tests made with rosin and pitch are recorded under "Results With Pitch."

RESULTS WITH ROSIN AND PETROLEUM.

Rosin mixes with almost all of the crude oils and their residual products that were tested in the laboratory. It mixed readily with the rubbery asphalt B2, forming a still more rubbery mass, but did not unite at all with the Indian Territory asphalt B4. All of the oils and petroleums increased the toughness of the rosin, and the consistency of the resulting mixtures varied from that of wagon grease to that of sticky rosin. Five parts of rosin melted with 1 part of Kansas crude petroleum gave a tough mass.

A series of experiments was made with rosin and Kansas crude oil P3 and Indian Territory No. 6 coal. With 5 per cent of rosin and 1 per cent of P3 oil a soft briquet was obtained, which hardened after exposure. Six per cent of rosin and 1 per cent of P3 oil yielded a briquet that was slightly plastic, very tough and strong in a warm room, but brittle and somewhat crumbly in the cold. It required several hours to harden. Six per cent of rosin and $1\frac{1}{2}$ per cent of P3 oil gave a briquet that could be bent in the fingers, but was no stronger than one made with 5 per cent of rosin and 1 per cent of oil.

Another series of experiments was made with equal parts of rosin and petroleum residue P2, which made a very sticky, plastic mass. With 6 parts of rosin and 4 parts of the petroleum P2, a mass was obtained that became hard when plunged into cold water. With 4 parts of rosin and 1 part of the petroleum P2, a mass was obtained that was slightly brittle at ordinary temperatures, but melted below 212° F. These proportions were therefore considered the best, and the mixture was tried with the Arkansas No. 5 coal by using 4 per cent of rosin and 1 per cent of the petroleum P2 as a binder. The resulting briquets were the best that were made in the laboratory and were hard enough to dent wood. When thrown violently against a cement floor, they broke across the grain and without producing any dirt. This briquet also burned exceptionally well.

The Indian Territory No. 6 coal was tried with a mixture of 3 per cent of rosin and 2 per cent of petroleum P2, but the briquet was oily, soft, and plastic. With a mixture containing 5 per cent of rosin and 2 per cent of petroleum P2 an excellent briquet was obtained. With $4\frac{1}{2}$ per cent of rosin and $1\frac{1}{2}$ per cent of petroleum P2 a briquet was made that was practically as good and represented the best proportion to be used with the coals of the type of Indian Territory No. 6. This briquet was nearly as good as those made with 10 per cent of the California asphalt B1.

RESULTS WITH ROSIN AND LIME.

With lime and rosin alone 15 per cent of lime and 85 per cent of rosin was the proportion required for a complete reaction. Rosin with unslaked lime sets after melting. If slaked lime be used, the reaction occurs at 212° F., and the mixture is light gray. This cement will soften and become sticky only at the melting point of lead. On cooling it is brittle, but stronger than rosin. With slaked lime rosin reacts and gives a satisfactory mixture up to 3 parts of rosin and 1 part of slaked lime.

RESULTS WITH ROSIN, LIME, AND PETROLEUM.

Rosin, slaked lime, and the petroleum residue P2 were mixed together in the proportion of 3 parts of rosin to 1½ parts of slaked lime and 1 part of petroleum P2. This mixture was tough and could be pulverized. Indian Territory No. 6 coal with 7 to 9.6 per cent of this mixture was heated with steam and when compressed made a medium hard briquet. It was not, however, tough enough nor did it burn satisfactorily.

Another combination was tried by using 3 parts rosin to 1½ parts slaked lime and 2 parts of Kansas P3 oil. This made a granular, nonbinding mass. With 2 parts of rosin to 1 of slaked lime and 1 of the P3 oil, a sticky, oily mass was obtained, but it was not at all tough.

A combination of rosin and the residual Beaumont petroleum P2 was made considerably tougher by the addition of lime, and for this combination relatively more lime was needed than when rosin alone was used. The best combination obtained was 6 parts of rosin to 3 parts of slaked lime and 2 parts of petroleum P2.

With Kansas crude oil no positive results could be obtained. Six parts of rosin, 3 of lime, and 4 of the Kansas oil P3 gave a greasy, granular mass without any apparent binding properties. With 3 parts of oil instead of 4 there was obtained a sticky mass, which, however, had no strength.

RESULTS WITH ROSIN AND SULPHURIC ACID.

Rosin was tested with sulphuric acid, the dilute acid having no effect on the rosin. Concentrated acid reacted with it, evolving SO₂ and forming a stiff, bright-red paste. When the paste was added to water there was formed a pink, soapy precipitate. Lignite was tested with 10 and with 13 per cent of this mixture, and fair briquets were made in each case. These softened slightly on standing and disintegrated in the fire. They were of about equal quality and equivalent to a briquet made with 12 per cent of rosin.

PETROLEUM TESTS.

All of the crude oils and petroleums can be stiffened to some extent by being mixed with rosin or hard pitch. None of the petroleums and oils that have been tested, however, could be used alone in briquetting any of the coals. Most of them were too fluid and had little or no binding qualities.

TESTS WITH MOLASSES.

The molasses experimented with in the laboratory can be bought in any market. It was always used in combination with lime. Thick molasses was diluted with twice its weight of water, and the lime was slaked to a paste with twice its weight of water. When these two reagents were mixed they had no apparent reaction until they were dry. By using 1 part of water to 1 of molasses and 2 of lime there was no apparent excess of either, and the mixture set at once to a rather hard cement similar to plaster of Paris. A series of experiments was made with Arkansas No. 6 coal by using slaked lime equivalent to 5 per cent of unslaked lime, diluted molasses equivalent to 1 per cent of the undiluted, and 94 per cent of the coal. The briquets were somewhat friable when taken from the mold, but became hard after standing 6 hours, and in 24 hours they were hard and strong. When immersed in water, they remained intact for about 48 hours, when they began to crumble. With a mixture equal to 3 per cent of unslaked lime and 3 per cent of undiluted molasses with 94 per cent of the coal, a briquet was obtained that was rather soft when first taken from the mold, but became hard in 5 hours. It burned with intumescence. With 1 per cent of lime and 5 per cent of molasses, the resultant briquet hardened slowly and at the end of 48 hours was tough and elastic, but not hard. At the end of seven days it had become extremely hard. In breaking it had a lustrous fracture and powdered only slightly. When tested in water, it softened and weakened only after several days. This ratio, 5 of molasses and 1 of lime, gave the best results.

Another test was made by mixing dry 4 per cent of quicklime with the coal and then adding 3 per cent of the molasses that had been previously diluted. The briquets as taken from the mold were soft and crumbly, having little or no cohesion. A brown Texas lignite, one-half crushed to pass through a 10-mesh sieve and the other half to pass through a 20-mesh sieve, was dried and tested, with the molasses and lime in the ratio of 5 to 1. The lime was first slaked to produce a dry, white powder, and the molasses was diluted with an equal weight of water. The very fine slaked lime equivalent to 1 per cent of unslaked lime and the lignite were mixed dry and then the diluted molasses, equal to 5 per cent of the pure molasses, was thoroughly mixed, a little at a time, with the mixed lime and lignite,

enough extra hot water being added to thoroughly moisten the mass. The main difficulty was in making the lignite moist. With 5 per cent of molasses and 2 per cent of lime, the resultant briquet was just coherent, and many small intersecting cracks developed when the pressure was relieved. On standing 48 hours the cracks had not enlarged and the briquets were hard, brittle, glossy, and brown. With 7 per cent of molasses and 2 per cent of lime, the briquets were similar to the others when first taken from the mold, but became slightly harder and darker in 48 hours. With 7 per cent of molasses and 4 per cent of lime fewer cracks developed when the pressure was relieved, but the briquets were still soft. In 48 hours, however, they were noticeably stronger and blacker than those of either of the other two mixtures. With 10 per cent of molasses and 3 per cent of lime similar cracks developed on relief of pressure, and on standing 48 hours the briquets were as hard as those with 7 and 4 per cent, respectively. With 14 per cent of molasses and 4 per cent of lime the briquet obtained was fair when first taken from the mold. It was somewhat crumbly, however, and at the end of 48 hours showed little change. Its character was not changed by using the extreme pressure obtainable on the screw machine. With 12 per cent of molasses and 4 per cent of lime the briquet was better, but still not strong while fresh. In 48 hours it had hardened somewhat, but had developed many of the drying cracks. With 16 per cent of molasses and 2 per cent of lime there was an excess of molasses, this being squeezed out under pressure. This briquet was fairly good while fresh, but became too brittle on drying. With a higher percentage of lime there was less tendency to crack, and although in the present series of experiments little lime was used in excess of the proportion required for forming calcium saccharate, it may be that a further addition of lime would make a briquet that would behave better in the fire.

TESTS WITH WAX TAILINGS.

A sample of wax tailings received from the Standard Oil Co. was combined with Arkansas coal. With 98 per cent of coal and 2 per cent of wax no coherence was established under moderate pressure. With 4 per cent of wax, either hot or cold, an elastic briquet was obtained which was fairly clean, but yielded under slowly applied pressure. The briquets were tested in a muffle and seemed to burn perfectly. With additional pressure a briquet (containing 4 per cent of wax) was obtained which, although it had no greater strength than the other; was smooth and clean. With 5 per cent of the wax and with light pressure, a briquet was obtained that was rather sticky and showed an excess of wax by sticking to the plunger.

The wax was also tried with some of the harder pitches. With 1 per cent of wax and 3 per cent of pitch X a noncoherent briquet was obtained; 1 per cent of wax and 5 per cent of pitch X produced the

same result, regardless of the pressure used; and with 2 per cent of wax and 4 per cent of pitch X a briquet was obtained similar to the one in which 13 per cent of pitch X alone was used. It was, however, not a satisfactory product.

TESTS WITH ACID SLUDGE.

Some acid sludge was received from the Gulf Refining Co., of Port Arthur, Tex. It was mixed with water, producing a weak acid solution, which did not seriously attack iron. Part of the sludge was a thin, greasy mass, and the remainder was rubbery and granular. The two parts did not mix when they were heated to the melting point. The stiffer part of the sludge did mix with melted rosin, and this mixture resembled the rubbery asphalt B2.

TESTS WITH LIME.

No experiments were made with lime alone, as the percentage required for making a coherent briquet was so high as to preclude entirely its use as a binder.

TESTS WITH CLAY.

Clay alone was tried up to 8 per cent, but in no case was a briquet obtained that would hold together on drying, although the clay used was a superior potter's clay.

COALS USED IN TESTS.

TESTS WITH COLORADO NO. 1.^a

Colorado No. 1 coal without previous drying was tried with pitch D. With 7 per cent of the pitch the briquet showed little cohesion; with 9 per cent a fair briquet was obtained; and with 10 per cent a good one was made, which stood up fairly well in the fire. With 11 per cent the briquet was better than with 10, but it was decided to use 10 per cent in making a run on the English machine.

TESTS WITH TEXAS NO. 1.^b

On account of the large areas of Texas No. 1 lignite and the importance of utilizing it as a fuel, many experiments were made in the laboratory in the effort to devise, if possible, a practical binder that could be used for briquetting it. It was first tried with pitch alone and then with various combinations, the results of the tests being largely unfavorable.

Results with pitch binder.—The lignite without previous drying was first tried with pitch B; it was crushed so that one-third was 40 mesh, another third 20 mesh, and the last third 10 mesh. With 6 per cent of pitch B heated in an assay crucible, without the addition of any water to make steam, a noncoherent briquet was obtained.

^a See table, p. 37.

^b See table, p. 40.

Some of the lignite dry was briquetted with 6 per cent of pitch B, without steam. The resultant briquet was noncoherent. With steam, however, a slightly better briquet was made. With 10 per cent of pitch B and steam a briquet was obtained that was fair when taken from the mold, but when tested in the fire, fell to pieces. With 12 per cent of pitch B a briquet was obtained that was also fair when fresh from the mold, but this also disintegrated immediately when placed in the fire. Kansas crude petroleum P3 was added to the lignite-and-pitch mixture without beneficial results.

Results with asphaltic pitch and rosin.—The next experiments were with asphaltic pitch B3 and rosin. With 3 per cent of pitch, 3 per cent of rosin, and 94 per cent of dry lignite, molded under very high pressure, a briquet only just coherent was obtained. With 4 per cent of this pitch and 4 per cent of rosin a fair briquet was obtained. Five per cent of the pitch and 5 per cent of rosin gave a fairly strong briquet, which crumbled slightly. With 6 per cent of each a briquet was obtained that was satisfactory physically, but crumbled in the fire. Ten per cent of pitch and 2 per cent of rosin, or 2 per cent of pitch and 10 per cent of rosin, made fair briquets, but not equal to the 6 to 6. All of the briquets made with the asphaltic pitch and rosin crumbled in the fire.

Results with rosin and lime.—Another series of experiments was made on this lignite with rosin and lime. With 6 per cent of rosin, 2 per cent of quicklime, and 92 per cent of the dry lignite, heated with steam, a briquet was obtained that had little or no coherence. With a mixture of 8 per cent of rosin and 88 per cent of lignite, to which was added 4 per cent of lime that had been boiled in water, a fair briquet was obtained. If this mixture is quite dry and is squeezed immediately on mixing, it will stand up in the fire better than a briquet made with rosin alone, but it is too soft to be handled.

Another sample of the lignite was crushed to 10 and 20 mesh fineness and heated until it was charred black. Twelve per cent of rosin and 2 per cent of lime, mixed in boiling water, when combined with the lignite, made a briquet that was excellent physically but disintegrated in the fire. Briquets made of the lignite, previously mixed dry with 12 per cent of rosin and 2 per cent of quicklime and heated with steam, cracked through the center when the pressure was relieved, and fragments tested in the fire disintegrated completely. The charred lignite was then ground to 60-mesh fineness and tested with 10 per cent of rosin and 2 per cent of lime, but there was no coherence at all in the briquet. With 12 per cent of rosin and 2 per cent of lime a poor briquet was obtained from this finely powdered material—one much inferior to that made by using the coarser charred lignite and the same ratio of rosin and lime. It also disintegrated completely in the fire.

It was thought that perhaps by mixing a bituminous coal with the lignite the binder would be more effective in holding the particles of lignite together. A mixture containing 10 per cent of Pocahontas coking coal, 8 per cent of rosin, 2 per cent of lime, and 80 per cent of the Texas lignite was briquetted. A briquet was obtained that was physically satisfactory, but it did not stand up in the fire any better than the others.

Results with lime.—The Texas lignite with 8 per cent of lime alone and steam made a briquet that was good when first taken from the mold, but on drying it began to crumble, indicating that lime, except when used in very large quantity, would not successfully briquet this lignite.

Results with asphalt.—Another series of experiments was made with the rubbery asphalt B2. With 12 per cent of the asphalt an excellent briquet was obtained by heating it with the lignite and without steam. The briquet was smooth and hard, but in the fire acted like the lime-rosin briquets and crumbled. With 12 per cent of the asphalt heated with steam the briquet was also physically good, and when the mold of the press was warm the resultant briquet was black and glossy. With 10 per cent of the asphalt and steam the briquet was just as good. It did not, however, stand up well in the fire. With 6 per cent of asphalt and 4 per cent of the asphaltic pitch B3 and steam the resultant briquet was almost noncoherent while warm, but became stronger on standing one hour. It was a fair briquet, but disintegrated in the fire.

Results with petroleum and rosin.—The lignite was also tested with rosin and the petroleum P2. With 9 per cent of rosin and 3 per cent of petroleum the briquets expanded immediately upon relief of pressure and large cracks developed. The cracks closed as the briquet cooled and the result was a briquet of fair hardness and strength; however, it disintegrated in the fire. With 7 per cent of rosin and 5 per cent of petroleum P2 the resultant briquet was too soft, showing the need of more rosin in the mixture. Some cracks developed, as in the previous briquets, and in burning, the briquet disintegrated completely. With 9 per cent of rosin, 3 per cent of petroleum P2, and 3 per cent of slaked lime, a good briquet was obtained that did not expand and crack upon relief of pressure. It burned in the fire with some cohesion, but was no better than the rosin and lime alone, and there was no apparent advantage in adding the petroleum.

TESTS WITH WEST VIRGINIA NO. 3.^a

West Virginia No. 3, a good coking coal, was tried in the laboratory to determine what could be done toward briquetting it without any binder whatever. Heated to 212° F. and then subjected to the maximum pressure on the hand press, a fair briquet was obtained without any binder. It was not strong, however, and caked in burning.

When the coal was heated until it began to smoke, beautiful lustrous briquets were obtained, which were little stronger than the previous ones. When heated until caking began and then subjected to the maximum pressure an excellent briquet was obtained.

TESTS WITH OTHER COALS.

Many other coals were tested in the laboratory and it was found that in nearly all cases a coking coal could be briquetted without the introduction of any binding material. In these experiments it was noted that the coal needed to be heated until it began to cake and that the hot coal had to be fed direct to the press, as even slight cooling seemed to diminish its briquetting properties. Another peculiarity indicated by a number of experiments was that after the coal had been heated until it began to cake and was then cooled and reheated it lost some of its briquetting properties.

BINDERS USED IN LATER ST. LOUIS AND NORFOLK TESTS.

In view of European tests and the results of previous experiments at St. Louis no binder was investigated except two of the pitches derived from the distillation of the tar obtained as a by-product in the manufacture of illuminating gas, namely, water-gas pitch (abbreviated to W. G. P.) and coal-tar pitch (abbreviated to C. T. P.). In order to obtain data that would be comparable, all briquets in any one lot were made with 6 to 8 per cent of the same binder.

The "flowing test" used at St. Louis for determining the hardness of the pitch was made in the following manner: The bulb of the thermometer was covered with a thin layer of melted pitch, which was allowed to harden. The thermometer was passed through a cork fitted in the mouth of a test tube, so that the bottom of the thermometer bulb was held 2 inches from the bottom of the tube. The test tube was then held in a vertical position in a beaker of boiling water. As the pitch became soft it gradually dropped from the thermometer in the form of a thread, and the temperature recorded when this thread reached the bottom of the test tube was called the "flowing point."

A method of determining the melting point of pitches that was used in testing some of the binders is more practical than the method given above. It is an English method, devised by F. G. Holmes,^a the details being as follows:

Several pieces of pitch are selected from different parts of the sample and cut into $\frac{1}{4}$ -inch cubes. The cubes are supported on 20-gage wires by heating the ends of the wires and pressing them into the pieces of pitch. They are then suspended in a 500-c. c. beaker of water, and heated by any convenient means at a uniform

^a Thorpe, Edward, Dictionary of applied chemistry, 1913, vol. 4, p. 290.

rate of 5° C. (9° F.) per minute. The cubes are placed on a level with the bulb of the thermometer; as the temperature rises the pieces of pitch are removed from time to time and twisted or squeezed with the fingers and the temperatures noted at which they become:

(1) Soft, so that the pitch can be readily twisted around several times.

(2) Very soft, so that the pitch yields to a very light pressure of the fingers.

(3) Fused, so that the pitch melts off of the wire.

Soft pitch softens at 104° and melts at 140° F. Moderately hard pitch softens at 140° and melts at about 176° F. Hard pitch softens at 176° and melts at about 248° F. For this type of pitch it is necessary to use a bath of oil or glycerin in place of the bath of water in which to heat the cubes.

A fractional distillation of the pitch was made as another verification of its quality. From this distillation the oils coming off under 572° F., between 572° and 680° F., and between 680° and 743° F. were recorded. Pitch that at the temperature of the mouth could be bitten nearly through before breaking proved the most satisfactory binder for most coals briquetted. The results of the tests of binders used in the later briquetting tests at St. Louis and Norfolk are given in the following table:

Tests of pitches used at St. Louis and Norfolk.

Laboratory No.	Lot No.	Extracted by carbon bisulphide.	Heat value.	Free carbon.	Flowing point.	Distillation test.				Kind of pitch.
						Oils up to 572° F.	Oils from 572° F. to 680° F.	Oils from 680° F. to 743° F.	Total oils up to 743° F.	
			<i>B. t. u.</i>	<i>P. ct.</i>	<i>° F.</i>	<i>P. ct.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	
2735		58.56		41.44	187	(a)	(a)	(a)	(a)	C. T. P.
2729		61.20		38.80	255	(a)	(a)	(a)	(a)	C. T. P.
2748		82.20		10.70	144	(a)	(a)	(a)	(a)	W. G. P.
2835		64.20		35.70	176	(a)	(a)	(a)	(a)	C. T. P.
3258	1	80.27	16,373	19.73	179	3.66	13.20	15.23	31.19	W. G. P.
3257	2	84.10	16,137	15.90	163	.76	11.26	12.78	24.83	W. G. P.
3296	3	82.43	16,427	17.58	171	2.00	14.0	8.50	24.60	W. G. P.
3293	4	63.64	15,941	36.36	165	3.08	11.57	11.82	26.47	C. T. P.
3410	5	79.98	16,478	20.02	175	.85	15.21	9.11	25.11	W. G. P.
3485	6	85.57	16,407	14.43	187	1.75	6.00	17.00	24.75	W. G. P.
3622	7	69.26	16,027	30.74	165	.85	11.15	10.15	22.15	C. T. P.
3624	8	89.60	16,193	10.40	172	1.25	10.35	8.40	20.00	W. G. P.
3692	9	69.71	16,103	30.29	158	1.89	10.90	10.08	22.82	C. T. P.
3885	10	95.20	16,870	4.80	140	1.43	14.83	18.18	34.44	W. G. P.
3962	11	77.79	16,198	29.21	159	3.00	12.75	10.75	26.50	W. G. P.
4120	12	97.70	17,090	2.30	154	.75	11.25	17.25	29.25	W. G. P.
4318	13	90.42	16,744	9.58	165	1.00	11.10	13.50	25.60	W. G. P.
4319	14	66.25	16,139	33.75	158	2.75	10.35	9.90	23.00	C. T. P.
4543	15	99.66	16,969	.36	144	1.62	17.31	20.12	39.05	W. G. P.
4683	16	89.31	16,637	10.69	162	1.28	11.64	10.06	23.98	W. G. P.
4685	17	90.56	16,576	9.44	154	1.61	9.66	14.30	25.47	W. G. P.
4806	18	96.90	16,864	3.10	144	1.09	9.44	15.23	25.76	W. G. P.
4672	19	94.50	16,905	5.50	126	1.33	16.53	15.15	39.01	W. G. P.
4825	20	100.00	17,156	.0	115	.46	1.05	4.25	5.76	(b)
5458	21	92.92	16,893	7.08	189	.88	12.34	22.44	35.66	W. G. P.
5563	22	90.33	16,718	9.67	201	1.45	10.32	11.80	23.07	W. G. P.
5639	23	98.13	16,781	1.87	194	1.87	4.21	14.54	20.62	W. G. P.
5641	24	98.44	16,978	1.56	196	1.71	14.05	16.20	31.96	W. G. P.
5940	26	96.60	16,790	3.40	178	1.27	9.88	17.75	28.90	W. G. P.
12009	28	64.24	15,608	35.76	158	24.0	6.19	3.63	33.82	C. T. P.

^a Data lost in fire.

^b Wax tailings.

Percentage of binder.—In handling a given sample of coal various percentages of binder were used in briquetting lots of 600 pounds. In all cases the fuel and the binder were weighed separately, and the percentages of binder given are taken from these figures. The extraction analyses by carbon bisulphide, the results of which for binders are given in the above table, were made from carefully selected samples of coals, pitches, and briquets, and the percentage of binder in the finished product was calculated from these determinations.*

The flour used as a binder at Norfolk (laboratory No. 6110) had the following approximate analysis:

	Per cent.
Moisture.....	13.30
Volatile combustible.....	72.30
Fixed carbon.....	13.50
Ash.....	.90
Sulphur.....	.10

Its heating value was 6,998 British thermal units.

OTHER BINDER INVESTIGATIONS.

In the laboratory investigations of J. E. Mills the substances named below were tested as binding materials in the manufacture of briquets.

INORGANIC BINDERS.

Some inorganic materials used as binder are: Clay, lime, magnesia, magnesia cement (magnesia oxide and magnesium chloride), plaster of Paris, Portland cement, natural cement, slag cement, water glass.

ORGANIC BINDERS.

Some of the organic materials used as binders are:

(1) Wood products, such as rosin, pitch (rosin and tar), pine tar, hardwood tar, Douglas fir tar, wood pulp, sulphite liquor (from paper mills).

(2) Sugar-factory residues, such as beet pulp, lime cake, beet-sugar molasses, cane-sugar molasses.

(3) Starch, such as corn starch, potato starch.

(4) Slaughterhouse refuse.

(5) Tars and pitches from coal, such as blast-furnace tar, producer-gas tar, illuminating-gas tar (from coal), by-product coke-oven tar, coal-tar creosote, various grades of pitches from various tars.

(6) Natural asphalts, such as impsenite, gilsonite, maltha, refined Trinidad, refined Bermuda, hard and refined or gum (from impregnated sandstone, etc.).

(7) Petroleum products, such as crude oil, residuum (asphalts, etc.), water-gas tar, water-gas tar pitch, wax tailings, acid sludge, asphalt tar, Pintsch-gas tar, Pittsburgh flux.

* See description of extraction test under heading "Tests Applied to Briquets."

CHARACTER AND RESULTS OF THE INVESTIGATIONS.

These investigations related not only to the nature and the amount of the binder necessary for making satisfactory briquets with each of the several coals tested, but also to the extent to which the binding quality of certain of these materials might be improved by the admixture of another binding material or another variety of coal.

Unless otherwise stated, 20 grams of each coal was mixed with the different percentages of the binding material and placed in a Batterssea crucible. A small proportion of water was then added and the mixture heated with sufficient stirring to thoroughly mix the binder and the coal until steam came off freely and only a small proportion of water was left. A part of the mixture, while still hot, was pressed in a small laboratory hand press, at a pressure of 3,500 to 4,000 pounds per square inch. Each briquet weighed approximately 5 grams.

The use of inorganic binding materials is not likely to prove practicable under ordinary conditions, except when coking coals can not be obtained to mix with noncoking coals. A small percentage of inorganic materials (such as magnesium oxide or carbonate, or plaster of Paris), if added to other binding materials, may cause the briquets to hold together better in the fire.

Of the more specific results in the testing of different binding materials the following tentative statements are made, pending further investigations:

The use of clay, lime, and cement as binding materials was found entirely unsatisfactory, for the reason that they added largely to the ash constituent of the briquet. The briquets made with these binders disintegrated on exposure to water and weather. To waterproof them by soaking in oils, etc., was found difficult and expensive. Water glass (or sodium silicate) was also unsuitable for use in this connection.

Briquets containing 4 to 6 per cent of magnesium oxide used as a binder held together satisfactorily in dry weather and in the fire, but they disintegrated on exposure to rainy weather or when immersed in water.

In the tests with 2 to 12 per cent of plaster of Paris used as a binder, the briquets made were hard but brittle, and quickly disintegrated. Three per cent of magnesia mixed with 6 to 8 per cent of water-gas tar pitch seemed to make a stronger briquet than the same percentage of pitch used alone; but the improvement in the quality of the briquets is not considered sufficient to cover the additional cost of the magnesia and the additional percentage of ash that it brings into the briquet.

None of the sugar-factory residues was considered satisfactory as a binding material. The briquets made with them disintegrate on

exposure to the weather, and no cheap waterproofing has as yet proved satisfactory.

None of the wood products, when used alone, was regarded as satisfactory. Some of these materials used in combination with other binders gave results of some promise and deserve further investigation.

The tests made using 0.5 to 3 per cent of starch as a binding material with different coals gave strong briquets. They were smokeless and held together in the fire until completely consumed; but these briquets went to pieces when wet or exposed to the weather for a considerable period of time. Experiments as to the possibility of cheaply waterproofing the briquets bound with starch, so as to make them hold together when exposed to water, were sufficiently successful to warrant further investigation in this direction. Starch is obtainable in large quantities and can be produced in almost any part of the country. Crude starch can probably be obtained at less than \$20 per ton. At this price the use of 1 per cent of starch as a binder would add only 20 cents per ton to the cost of the briquets.

The scarcity and high price of slaughterhouse refuse would prohibit its use as a binding material for briquets. Satisfactory results were not obtained in the tests.

The tests with coal tars and the different grades of pitch made from those tars indicated that probably in the pitches the most satisfactory binders for the manufacture of briquets will be found; and that these can be made at a price that will bring the cost of the binding material used to not more than 50 to 75 cents per ton of briquets. Briquets made from a majority of the bituminous coals with a good grade of pitch as a binder burn well and are sufficiently strong to bear ordinary handling. They will stand exposure to the weather for a number of years without serious deterioration. The supply of pitch is large, and can easily be increased. The price now ranges from \$9 to \$11 per ton. The pitches obtained from different plants and those obtained at different times from the same plants vary considerably, not only in their boiling points, but in other respects; and in the case of similar pitches the percentage necessary for the proper working of different coals seemed to vary according to differences in the quality of the coals. It was found to be better to mix with a noncoking coal 10 to 20 per cent of a coking coal, rather than to increase the percentage of pitch.

In the investigation of the asphalts as binding materials, impsonite from the Indian Territory (Oklahoma) was rather unsatisfactory. In a number of tests with noncoking coals the result was improved by the addition to such coal of 5 to 10 per cent of impsonite in addition to 3 to 5 per cent of ordinary pitch or some other binding material. Four to eight per cent of gilsonite and other asphalts from Utah gave

fairly satisfactory results as binders. These materials are found in Utah and elsewhere in large quantities. The price at present is too high to permit their extensive use.

Experiments with several other asphaltic materials were not altogether satisfactory. Asphaltic tar yielded fairly good results as a waterproofing material in briquets made with starch.

Four to six per cent of maltha (a liquid asphalt) used as a binder yielded fairly satisfactory results with coking coals. With noncoking coals it was insufficient.

Crude petroleum has been tested as binding materials, with satisfactory results. Six to eight per cent of asphaltic petroleum was used successfully in waterproofing briquets made with a starch binder. It is doubtful whether such use would prove entirely satisfactory on a commercial scale.

The petroleum residuums proved to be successful binding materials somewhat in proportion as the percentage of asphalt in them increased. That obtained from the Gulf Refining Co., of Port Arthur, Tex., melted at a temperature of 212° F., and about 99.38 per cent was soluble in carbon bisulphide. Another sample of asphaltic petroleum from the Gulf Refining Co. flowed at a temperature of 203° F., and carbon bisulphide dissolved 99.88 per cent. Four to six per cent of this material gave excellent results as a binder for briquets. Water-gas tar, which is obtained from illuminating-gas plants, was not tested sufficiently to give satisfactory results. It is necessary, after this material is mixed with the coal, to raise its temperature sufficiently high to liquefy and perhaps even vaporize it.

The water-gas pitch furnished by the Barrett Manufacturing Co. flowed at a temperature of 198° F. Four and one-half to seven per cent of this pitch is equivalent to 6 to 8 per cent of coal-tar pitch, and makes satisfactory briquets. The cost of this binder would be from 45 to 65 cents per ton of briquets.

The wax tailings used in these investigations were obtained from the Standard Oil Co. They melt at about 158° F. Four to six per cent of this material produces good briquets. The quantity of this material produced in the United States is small, and the price is about 6 cents per gallon. This price would make the binding material used in a ton of briquets cost approximately 45 to 60 cents.

Tests made with acid sludge as a binder were not satisfactory. The sludge not only added the unwelcome sulphur, but its binding qualities were inferior.

Asphalt tar was unsatisfactory as a binder, even when 8 to 12 per cent was used, inasmuch as the briquets fell to pieces in the fire. Its use for waterproofing, when starch or other material had been used as a binder, proved fairly satisfactory.

Pintsch-gas tar was obtainable in so small a quantity that only preliminary tests were made covering its use as a binder, and they were only partly satisfactory.

In making coal briquets binding is most successful when the particles of coal are coated and when the void spaces are filled with the binding material. This is best accomplished when the temperature of the mixture, before compression, is raised sufficiently to liquefy or vaporize the binder. The relation between the coal and the binder seems to be physical rather than chemical, though it is possible that certain minor chemical changes may accompany the briquetting operation.

The quantity of the binding material necessary will therefore depend on the aggregate of the surfaces of the coal particles to be coated, on the void spaces to be filled, and on the general physical and chemical character of the coal. Coking coals require less binder than noncoking coals. The percentage of binder necessary for noncoking coals may be diminished and the quality of the briquet improved by the addition of 10 to 20 per cent of coking coal.

Furthermore, when coal is finely pulverized the briquetting is facilitated by the mixture of a considerable percentage of the same or another coal—preferably a nonslacking coal—crushed to sizes ranging from $\frac{1}{8}$ to $\frac{1}{4}$ inch.

The following are mentioned as the more important requisites of a suitable binding material for use in the manufacture of briquets:

(1) It must be inexpensive, on account of the small difference between the prices of slack or fine coal and lump coal.

(2) It should be capable of abundant production in different parts of the country, in order to avoid the necessity of long transportation.

(3) It should be of such character as to make it easily handled and applied at workable temperatures.

(4) It should hold the briquet together, not only during ordinary handling and transportation, but also during protracted exposure to the weather and while burning.

(5) It should not add appreciably to the ash of the coal nor increase the clinker formation in the ash. It should not give off fumes nor seriously increase the smoke in the burning of the briquets. Binding materials that have a low melting or boiling temperature give off more smoke because of the fact that this smoke originates at a temperature too low for complete combustion.

(6) It should increase the heating quality of the coal used in the manufacture of the briquets.

TESTS OF AMERICAN LIGNITE IN GERMANY.

In August, 1904, under the supervision of J. A. Holmes, samples of lignite from Lehigh, N. Dak., and from near Rockdale, Tex., were shipped to Magdeburg and Zeitz, Germany, to determine the

feasibility of briquetting these lignites with the German presses. The tests conducted by Dr. Holmes were successful, and the briquets made with these lignites subsequently stood the tests applied to the German lignite briquets, which are extensively used for domestic purposes.

To a part of the briquets made at Magdeburg 2 per cent of ordinary coal-tar pitch was added, with the view of determining what, if any, advantage would result from such an addition. Laboratory tests indicated that the briquets to which this pitch was added burned more rapidly, with more flame, and at a higher heat efficiency than those without it, and the North Dakota lignite seemed to give off fewer sparks in burning. It is estimated that the cost of manufacturing these briquets (not including the cost of the lignite) should not exceed 50 cents per ton. The analyses of the lignite briquets made at Magdeburg are as follows:

Analyses of briquets made at Magdeburg, Germany (lignite dried before briquetting).

	From Lehigh, N. Dak.		From Rockdale, Tex.	
	Without pitch or other binder.	With 2 per cent of pitch added.	Without pitch or other binder.	With 2 per cent of pitch added.
Proximate:				
Moisture.....	12.96	11.92	8.88	8.92
Volatile matter.....	37.79	38.68	40.92	41.33
Fixed carbon.....	38.63	41.11	41.90	42.35
Ash.....	10.62	8.99	8.30	7.40
Sulphur.....	.75	1.04	.97	1.07
Ultimate:				
Hydrogen.....	4.99	5.09	5.21	5.23
Carbon.....	52.88	56.26	59.06	59.61
Nitrogen.....	.65	.73	1.15	1.14
Oxygen.....	30.11	27.89	25.31	25.55
Calories.....	4,941	5,332	5,682	5,828
British thermal units.....	8,804	9,598	10,228	10,490
Moisture in undried sample of lignite as received at Magdeburg.....	28.87		32.07	

LABORATORY INVESTIGATIONS OF BINDERS AND FUELS.

During the past three years the author has carried on at Pittsburgh laboratory briquetting investigations of certain binding materials, using the hydraulic hand press previously described. As the work progressed, it was found desirable to record data that were not considered essential in the earlier experiments; hence data for the different tests are not alike complete. These laboratory tests should not be confused with the tests made on the large machines. No attempt was made to prepare briquets in sufficient quantity for combustion tests, as the object of the tests was merely to study the briquetting qualities of various fuels and binders.

METHOD OF MAKING LABORATORY BRIQUETTING TESTS.

In making a laboratory test when the mixture was heated by steam, 1,000 grams of a mixture of fuel and binder was prepared, using the proper proportion of the binder. If a solid binder was used, it was first crushed to pass a 20-mesh sieve. If liquid or viscous binders were being tested, they were warmed sufficiently to cause them to mix thoroughly with the coal. The fuel and the binder in either case were mixed by kneading and stirring until a uniform mixture was obtained. Two hundred grams of this mixture was placed in a 2-quart pan and live steam (as dry as possible but not superheated) was blown into the mixture through a $\frac{1}{4}$ -inch pipe bent at right angles and attached to a flexible rubber hose. This heating was carried on for periods of 20, 30, 60, 90, or 120 seconds, as found advisable from previous experiments with fuels of similar type. After weighing to determine the gain in moisture, the mixture was transferred to the mold of the hydraulic press and the temperature noted. A reading was made on the plunger to determine the length of the mixture before compression. Pressure was then applied and registered on a large gage. When the desired pressure had been obtained, another plunger reading was made, and the difference between this and the reading before compression gave the reduction of volume.

DESIGNATED QUALITIES OF BRIQUETS.

The values of the letters A, B, C, D, or E, used in the tables to designate the quality of fresh briquets and of air-dried briquets, are as follows:

A: An excellent briquet, difficult to fracture with the hands, and having smooth, firm surfaces and sharp, strong edges.

B: A good briquet, lacking some of the qualities mentioned under A, but yet a satisfactory and practical briquet.

C: A fair briquet, somewhat lacking in strength and somewhat rough and soft on the surface and with edges that crumble easily, so that it is hardly a satisfactory briquet.

D: A briquet of poor quality and entirely unsatisfactory, lacking strength, smooth surface, and firm edges.

E: A briquet so poor that it crumbled to pieces when removed from the mold.

The abbreviation "S. J." in the table under the item "Method of Heating" indicates the use of a steam jet. The letters W. G. P. represent water-gas pitch.

COMPRESSION TEST.

As a measure of the strength, the compressive strength or the amount of pressure the laboratory briquets would stand without breaking was more closely comparable than that noted in briquets made on the larger presses.

DETAILS OF LABORATORY BINDER TESTS.

In the following table are given the physical and chemical characteristics of the binders used in the laboratory tests:

Binders used in laboratory briquetting tests.

Lot No.	Laboratory No.	Kind of binder.	Ex- tracted by carbon bisul- phide.	Heat value.	Flow- ing point.	Proximate analysis.				
						Mois- ture.	Vola- tile com- bustible matter.	Fixed carbon.	Ash.	Sul- phur.
26	5940	Water-gas pitch.....	<i>Per ct.</i>	<i>B. t. u.</i>	<i>° F.</i>					
27	9298	Cell pitch.....	96.60	16,780	178					
29	11990	Hard wood-tar pitch.....		6,550	(a)	8.10	57.79	9.09	25.04	2.21
31	14798	Wheat flour.....		14,233	82	1.20	78.87	19.67	.26	.14
32	12515	Sulphite liquor.....		6,856		13.83	78.18	7.99	.0	.12
34	14769	do.....		4,596		39.16	43.04	8.69	9.11	2.13
				4,734		37.98	38.59	13.52	9.91	2.06

a Burned to ash without melting.

TESTS WITH PITTSBURGH NO. 15.^a

The data obtained from laboratory tests of Pittsburgh No. 15 and various binders are given in the following tables:

Laboratory briquetting tests of Pittsburgh No. 15 without binder.

	Test No.						
	L 158.	L 169A.	L 169B.	L 169C.	L 169D.	L 169E.	L 169F.
Size as used:							
Over $\frac{1}{8}$ inch, per cent.....	0.8	0.8	0.8	0.8	0.8	0.8	0.8
$\frac{1}{8}$ to $\frac{1}{4}$ inch, per cent.....	45.3	45.3	45.3	45.3	45.3	45.3	
$\frac{1}{4}$ to $\frac{3}{8}$ inch, per cent.....	32.4	32.4	32.4	32.4	32.4	32.4	
$\frac{3}{8}$ to $\frac{1}{2}$ inch, per cent.....	11.7	11.7	11.7	11.7	11.7	11.7	
Through $\frac{1}{2}$ inch, per cent.....	9.8	9.8	9.8	9.8	9.8	9.8	9.8
Briquetting temperature, °F.	Warm.	Cold.	182	178	192	Cold.	Cold.
Binder, amount used, per cent.	None.	None.	None.	None.	None.	None.	None.
Method of heating used.	S. J.	Cold.	S. J.	S. J.	S. J.		
Weight of:							
Mixture used, grams.....	200	200	200	100	200	100	100
Briquet, average, grams.....	200	200	206			100	100
Condition of fuel, dried or un- dried.	Undried.	Dried.	Dried.	Dried.	Dried.	Dried.	Dried.
Pressure used, pounds per square inch.....	14,000	14,000	14,000	30,000	6,000	30,000	20,000
Moisture:							
In fuel, per cent.....	30.7	10.0	10.0	10.0	10.0	10.0	10.0
Added in heating, per cent.....			4.0	3.0	5.0	0	0
In briquet, per cent.....			14.0	13.0	15.0	10.0	10.0
Length of heating, minutes.....				$\frac{1}{2}$	$\frac{1}{2}$		
Reduction of volume by pres- sure square inch.....			46				
Quality of briquet:							
Fresh ^b	E	D	D	D	E	D+	D+
Air-dried ^b		D	D	D	E	D+	D+

^a See table, p. 39.

^b See p. 71 for value of letters.

Laboratory briquetting tests of Pittsburgh No. 15 with water-gas pitch binder.

	Test No.					
	L 160.	L 161.	L 162.	L 170.	L 171.	L 172.
Size as used:						
Over $\frac{1}{8}$ inch, per cent.	0.8	0.8	0.8	0.8	0.8	0.8
$\frac{1}{8}$ to $\frac{1}{4}$ inch, per cent.	45.3	45.3	45.3	45.3	45.3	45.3
$\frac{1}{4}$ to $\frac{3}{8}$ inch, per cent.	32.4	32.4	32.4	32.4	32.4	32.4
$\frac{3}{8}$ to $\frac{1}{2}$ inch, per cent.	11.7	11.7	11.7	11.7	11.7	11.7
Through $\frac{1}{2}$ inch, per cent.	9.8	9.8	9.8	9.8	9.8	9.8
Briquetting temperature, °F.	156	156	172	186	177	175
Binder:						
Lot No.	26	26	26	26	26	26
Amount used, per cent.	4	6	8	4	6	8
Method of heating used	S. J.	S. J.	S. J.	S. J.	S. J.	S. J.
Weight of:						
Mixture used, grams.	200	200	200	200	200	200
Briquet, average, grams.	207	207	208	208	207	208
Condition of fuel, dried or undried.	Undried.	Undried.	Undried.	Dried.	Dried.	Dried.
Pressure used, pounds per square inch.	6,000	6,000	6,000	6,000	6,000	6,000
Moisture:						
In fuel, per cent.	30.7	30.7	30.7	10.0	10.0	10.0
Added in heating, per cent.	5.5	5.5	6.0	4.5	4.0	4.5
In briquet, per cent.	36.2	36.2	36.7	14.5	14.0	14.5
Length of heating, minutes.	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$
Reduction of volume by pressing, per cent.	45	45	45	45	45	45
Crushing strength, pounds per square inch.	200	200	200	200	300	650
Quality of briquet:						
Fresh.	D	D	D	D	D	C
Air-dried.	D	D	D	D	C	B

Laboratory briquetting tests of Pittsburgh No. 15 with wheat-flour binder.

	Test No.					
	L 166.	L 167.	L 168.	L 176.	L 177.	L 178.
Size as used:						
Over $\frac{1}{8}$ inch, per cent.	0.8	0.8	0.8	0.8	0.8	0.8
$\frac{1}{8}$ to $\frac{1}{4}$ inch, per cent.	45.3	45.3	45.3	45.3	45.3	45.3
$\frac{1}{4}$ to $\frac{3}{8}$ inch, per cent.	32.4	32.4	32.4	32.4	32.4	32.4
$\frac{3}{8}$ to $\frac{1}{2}$ inch, per cent.	11.7	11.7	11.7	11.7	11.7	11.7
Through $\frac{1}{2}$ inch, per cent.	8.8	8.8	8.8	8.8	8.8	8.8
Briquetting temperature, °F.	196	191	181	190	194	183
Binder:						
Lot No.	31	31	31	31	31	31
Amount used, per cent.	3	5	7	3	5	7
Method of heating used	S. J.	S. J.	S. J.	S. J.	S. J.	S. J.
Weight of:						
Mixture used, grams.	200	200	200	200	200	200
Briquet, average, grams.	208	211	210	210	211	211
Condition of fuel, dried or undried.	Undried.	Undried.	Undried.	Dried.	Dried.	Dried.
Pressure used, pounds per square inch.	6,000	6,000	6,000	6,000	6,000	6,000
Moisture:						
In fuel, per cent.	30.7	30.7	30.7	10.0	10.0	10.0
Added in heating, per cent.	7.0	7.5	7.5	5.5	7.0	6.5
In briquet, per cent.	37.7	38.2	38.2	15.5	17.0	16.5
Length of heating, minutes.	1	1	1	1	1	1
Reduction of volume by pressing, per cent.	42	43	40	46	46	46
Crushing strength, pounds per square inch.	600	1,000	1,125	750	1,150	1,100
Quality of briquet:						
Fresh.	C	A	A	B	A	aA
Air-dried.	B	A	A	B	A	aA

• Edges brittle.

Laboratory briquetting tests of Pittsburgh No. 15 with cell-pitch binder.

	Test No.				
	L 163.	L 164.	L 165.	L 174.	L 175.
Size as used:					
Over $\frac{1}{8}$ inch, per cent.....	0.8	0.8	0.8	0.8	0.8
$\frac{1}{8}$ to $\frac{1}{4}$ inch, per cent.....	45.3	45.3	45.3	45.3	45.3
$\frac{1}{4}$ to $\frac{3}{8}$ inch, per cent.....	32.4	32.4	32.4	32.4	32.4
$\frac{3}{8}$ to $\frac{1}{2}$ inch, per cent.....	11.7	11.7	11.7	11.7	11.7
Through $\frac{1}{2}$ inch, per cent.....	9.8	9.8	9.8	9.8	9.8
Briquetting temperature, °F.....	169	167	173	163	174
Binder:					
Lot No.....	27	27	27	27	27
Amount used, per cent.....	4	6	8	6	8
Method of heating used.....	S. J.	S. J.	S. J.	S. J.	S. J.
Weight of:					
Mixture used, grams.....	200	200	200	200	200
Briquet, average, grams.....	209	207	207	206	208
Condition of fuel, dried or undried.....	Undried.	Undried.	Undried.	Dried.	Dried.
Pressure used, pounds per square inch.....	6,000	6,000	6,000	6,000	6,000
Moisture:					
In fuel, per cent.....	30.7	30.7	30.7	10.0	10.0
Added in heating, per cent.....	6.0	5.5	6.0	4.0	5.0
In briquet, per cent.....	36.7	36.2	36.7	14.0	15.0
Length of heating, minutes.....	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$
Reduction of volume by pressing, per cent.....	43	43	45	50	50
Crushing strength, pounds per square inch.....	200	200		550	926
Quality of briquet:					
Fresh.....	D	D	D	C	B
Air-dried.....	D	C	C	C	B

The results obtained from these experiments show that North Dakota lignite (Pittsburgh No. 15) can not be successfully briquetted without binder, even under high pressure. The briquets made from the undried fuel with 8 per cent of the water-gas pitch were not satisfactory. When the fuel was dried to contain only 10 per cent of moisture, and 8 per cent of the water-gas pitch was used, a satisfactory briquet was obtained. With 3 to 5 per cent of flour as a binder excellent briquets were obtained from both the dried and the undried fuel, although they would not be waterproof and would have to be stored under cover.

With 8 per cent of cell pitch the dried fuel made a satisfactory briquet. Of course briquets made with this binder would not be waterproof and would have to be stored under cover. The source and analysis of this fuel are given under "Details of Briquetting Tests."

TESTS WITH PITTSBURGH NO. 17.^a

The data obtained from laboratory tests of Pittsburgh No. 17 and various binding materials are given in the following tables:

^a See table, p. 33.

Laboratory briquetting tests of Pittsburgh No. 17 without binder.

	Test No.						
	L 225.	L 226.	L 227.	L 228.	L 237.	L 238.	L 239.
Size as used:							
Over $\frac{1}{8}$ inch, per cent.	9.7	9.7	9.7	9.7	9.7	9.7	9.7
$\frac{1}{8}$ to $\frac{1}{4}$ inch, per cent.	40.5	40.5	40.5	40.5	40.5	40.5	40.5
$\frac{1}{4}$ to $\frac{3}{8}$ inch, per cent.	26.5	26.5	26.5	26.5	26.5	26.5	26.5
$\frac{3}{8}$ to $\frac{1}{2}$ inch, per cent.	12.7	12.7	12.7	12.7	12.7	12.7	12.7
Through $\frac{1}{2}$ inch, per cent.	10.6	10.6	10.6	10.6	10.6	10.6	10.6
Briquetting temperature, °F.	Cold.	Cold.	Cold.	157	189	194	187
Binder, amount used, per cent.	None.	None.	None.	None.	None.	None.	None.
Method of heating used.	Cold.	Cold.	Cold.	(e)	S. J.	S. J.	(e)
Weight of:							
Mixture used, grams.	200	200	155	150	200	150	150
Briquet, average, grams.	199	200	155	146	202	157	149
Condition of fuel, dried or undried.	Undried.	Undried.	Undried.	Undried.	Dried.	Dried.	Dried.
Pressure used, pounds per square inch.	14,000	6,000	32,000	32,000	14,000	32,000	32,000
Moisture:							
In fuel, per cent.	31.6	31.6	31.6	31.6	14.1	14.1	14.1
Added in heating, per cent.				-1.5	3.5	4.5	0
In briquet, per cent.	31.6	31.6	31.6	30.1	17.6	18.6	14.1
Length of heating, minutes.	Cold.	Cold.	Cold.	(b)	$\frac{1}{2}$	1	(b)
Reduction of volume by pressing, per cent.	50	48			43		
Crushing strength, pounds per square inch.						200	
Quality of briquet:							
Fresh.	D	D	D	E	C	D.	C
Air-dried.	D	D	D		C	D.	C

* Gas flame.

b Until hot.

Laboratory briquetting tests of Pittsburgh No. 17 with water-gas pitch binder.

	Test No.					
	L 199.	L 200.	L 201.	L 202.	L 203.	L 229.
Size as used:						
Over $\frac{1}{8}$ inch, per cent.	9.7	9.7	9.7	9.7	9.7	9.7
$\frac{1}{8}$ to $\frac{1}{4}$ inch, per cent.	40.5	40.5	40.5	40.5	40.5	40.5
$\frac{1}{4}$ to $\frac{3}{8}$ inch, per cent.	26.5	26.5	26.5	26.5	26.5	26.5
$\frac{3}{8}$ to $\frac{1}{2}$ inch, per cent.	12.7	12.7	12.7	12.7	12.7	12.7
Through $\frac{1}{2}$ inch, per cent.	10.6	10.6	10.6	10.6	10.6	10.6
Briquetting temperature, °F.	201	201	199	198	198	187
Binder:						
Lot No.	26	26	26	26	26	26
Amount used, per cent.	4	5	6	7	8	6
Method of heating used.	S. J.	S. J.	S. J.	S. J.	S. J.	S. J.
Weight of:						
Mixture used, grams.	200	200	200	200	200	200
Briquet, average, grams.	209	209	209	209	210	207
Condition of fuel, dried or undried.	Undried.	Undried.	Undried.	Undried.	Undried.	Dried.
Pressure used, pounds per square inch.	4,000	4,000	4,000	4,000	4,000	4,000
Moisture:						
In fuel, per cent.	31.6	31.6	31.6	31.6	31.6	15.7
Added in heating, per cent.	6.5	7.0	6.5	5.5	6.5	4.5
In briquet, per cent.	38.1	38.6	38.1	37.1	38.1	20.2
Length of heating, minutes.	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$
Reduction of volume by pressing, per cent.	47	47	46	46	46	49
Crushing strength, pounds per square inch.	-300	-300	-300	a -300	350	400
Quality of briquet:						
Fresh.	D	D	D	D	D	C
Air-dried.	D	D	D	D	C	C

a Indicates that strength was less than 300 pounds per square inch, the lowest pressure that the gage would register.

Laboratory briquetting tests of Pittsburgh No. 17 with flour binder.

	Test No.							
	L 204.	L 205.	L 206.	L 207.	L 208.	L 209.	L 210.	L 230.
Size as used:								
Over $\frac{1}{8}$ inch, per cent.	9.7	9.7	9.7	9.7	9.7	9.7	9.7	9.7
$\frac{1}{8}$ to $\frac{1}{4}$ inch, per cent.	40.5	40.5	40.5	40.5	40.5	40.5	40.5	40.5
$\frac{1}{4}$ to $\frac{3}{8}$ inch, per cent.	26.5	26.5	26.5	26.5	26.5	26.5	26.5	26.5
$\frac{3}{8}$ to $\frac{1}{2}$ inch, per cent.	12.7	12.7	12.7	12.7	12.7	12.7	12.7	12.7
Through $\frac{1}{2}$ inch, per cent.	10.6	10.6	10.6	10.6	10.6	10.6	10.6	10.6
Briquetting temperature, °F.	201	201	203	201	201	192	201	199
Binder:								
Lot No.	31	31	31	31	31	31	31	31
Amount of, per cent.	2	3	4	5	6	7	8	6
Method of heating used.	S. J.	S. J.	S. J.	S. J.	S. J.	S. J.	S. J.	S. J.
Weight of:								
Mixture used, grams.	200	200	200	200	200	200	200	200
Briquet, average, grams.	211	210	211	210	211	210	210	209
Condition of fuel, dried or undried.	Undried.	Undried.	Undried.	Undried.	Undried.	Undried.	Undried.	Dried.
Pressure used, pounds per square inch.	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000
Moisture:								
In fuel, per cent.	31.6	31.6	31.6	31.6	31.6	31.6	31.6	15.7
Added in heating, per cent.	7.5	7.0	7.5	7.0	7.5	7.0	7.0	6.0
In briquet mixture, per cent.	39.1	38.6	39.1	38.6	39.1	38.6	38.6	21.7
Length of heating, minutes.	1	1	1	1	1	1	1	1
Reduction of volume by pressing, per cent.	45	45	45	45	45	46	45	44
Crushing strength, pounds per square inch.	600	750	1,100	1,125	1,200	1,075	1,525	1,000
Quality of briquet:								
Fresh.	D	C	D	B	A	B	A	B
Air-dried.	C	B	B	A	A	B	A	B

Laboratory briquetting tests of Pittsburgh No. 17 with sulphite-liquor binder.

	Test No.							
	L 218.	L 219.	L 220.	L 221.	L 222.	L 223.	L 224.	L 231.
Size as used:								
Over $\frac{1}{8}$ inch, per cent.	9.7	9.7	9.7	9.7	9.7	9.7	9.7	9.7
$\frac{1}{8}$ to $\frac{1}{4}$ inch, per cent.	40.5	40.5	40.5	40.5	40.5	40.5	40.5	40.5
$\frac{1}{4}$ to $\frac{3}{8}$ inch, per cent.	26.5	26.5	26.5	26.5	26.5	26.5	26.5	26.5
$\frac{3}{8}$ to $\frac{1}{2}$ inch, per cent.	12.7	12.7	12.7	12.7	12.7	12.7	12.7	12.7
Through $\frac{1}{2}$ inch, per cent.	10.6	10.6	10.6	10.6	10.6	10.6	10.6	10.6
Briquetting temperature, °F.	199	197	201	201	196	196	198	
Binder:								
Lot No.	32	32	32	32	32	32	32	32
Amount used, per cent.	3	4	5	6	7	8	9	6
Method of heating used.	S. J.	S. J.	S. J.	S. J.	S. J.	S. J.	S. J.	S. J.
Weight of:								
Mixture used, grams.	200	200	200	200	200	200	200	200
Briquet, average, grams.	210	209	209	210	208	209	208	207
Condition of fuel, dried or undried.	Undried.	Undried.	Undried.	Undried.	Undried.	Undried.	Undried.	Dried.
Pressure used, pounds per square inch.	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000
Moisture:								
In fuel, per cent.	31.6	31.6	31.6	31.6	31.6	31.6	31.6	15.7
Added in heating, per cent.	6.0	6.0	6.0	6.5	5.5	6.0	6.5	5.0
In briquet mixture, per cent.	37.6	37.6	37.6	38.1	37.1	37.6	38.1	20.7
Length of heating, minutes.	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$
Reduction of volume by pressing, per cent.	45	46	46	46	46	46	46	43
Crushing strength, pounds per square inch.								450
Quality of briquet:								
Fresh.	E	E	E	D	D	D	D	D
Air-dried.	E	E	E	D	D	D	D	C

The North Dakota lignite represented by Pittsburgh No. 17 can be briquetted without binder by pressure alone if partly dried and heated with steam before pressing. The dried fuel that was briquetted without binder contained 14.1 per cent of moisture. The undried fuel was not satisfactorily briquetted without binder.

With 6 per cent of water-gas pitch better briquets could be made from the dried than from the undried fuel, whereas 8 per cent did not produce as good results with the undried fuel.

Four per cent of flour binder was required to make satisfactory briquets of this fuel in an undried condition, and 6 per cent of flour was sufficient to make excellent briquets with the undried fuel.

Nine per cent of sulphite liquor (lot No. 32) was insufficient to make fair briquets from this undried fuel, but 6 per cent made good briquets when the dried fuel contained 15.7 per cent of moisture.

The sample had the following analysis:

Proximate analysis (as received).

	Per cent.
Moisture	31.99
Volatile matter.....	31.70
Fixed carbon.....	31.27
Ash.....	5.04
Sulphur.....	.46
Heating value.....	B. t. u. 7,353

TESTS WITH PENNSYLVANIA ANTHRACITE CULM.

The data obtained from laboratory tests of Pennsylvania anthracite culm (Pennsylvania No. 3^a) and various binders are given in the following table:

Laboratory briquetting tests of Pennsylvania culm with cell-pitch binder.

	Test No.	
	L 80.	L 81.
Briquetting temperature, °F.....	176	203
Binder:		
Lot No.....	27	27
Amount used, per cent.....	3	3
Method of heating used.....	Gas flame.	S. J.
Weight of:		
Mixture used, grams.....	212	200
Briquet, average, grams.....	203	200
Condition of fuel, dried or undried.....	Undried.	Undried.
Pressure used, pounds per square inch.....	4,000	4,000
Moisture added in heating, per cent.....	4.5	4.5
Length of heating, minutes.....	46	44
Reduction of volume by pressing, per cent.....	46	44
Crushing strength, pounds per square inch.....	2,700	2,700
Quality of briquet:		
Fresh.....	B	E
Air-dried.....	A	A

Laboratory briquetting tests of Pennsylvania culm with sulphite-liquor binder.

	Test No.					
	L 77A.	L 77B.	L 77C.	L 78.	L 82.	L 83.
Briquetting temperature, °F..	Cold.	201	180	158	Cold.	Hot.
Binder:						
Lot No.....	34	34	34	34	34	24
Amount used, per cent.....	10	5	1	4.3	5	5
Method of heating used.....	Cold.	Gas flame.	Gas flame.	Heated in mold.	Cold.	Steam jet.
Weight of:						
Mixture used, grams.....	200	200	200	200	200	200
Briquet, average, grams.....	200	200	200	200	200	200
Condition of fuel, dried or undried.....	Undried.	Undried.	Undried.	Undried.	Undried.	Undried.
Pressure used, pounds per square inch.....	4,000	4,000	4,000	4,000	6,000	6,000
Length of heating, minutes.....	Cold.	Until warm.	Until warm.			1
Reduction of volume by pressing, per cent.....	40	40	40	44		42
Crushing strength, pounds per square inch.....	2,600	2,600				
Quality of briquet:						
Fresh.....	D	B	E	C	C	E
Air-dried.....	A	A	E	B	B	E

Laboratory briquetting tests of Pennsylvania culm with sulphite-liquor binder—Continued.

	Test No.		
	L 85.	L 84.	L 86. ^c
Briquetting temperature, °F.....	Hot.	176	Cold.
Binder:			
Lot No.....	34	34	34
Amount used, per cent.....	5	5	5
Method of heating used.....	Oven.	Gas flame.	Cold.
Weight of:			
Mixture used, grams.....	200	200	200
Briquet, average, grams.....	200	97.5	200
Condition of fuel, dried or undried.....	Dried.	Dried.	
Pressure used, pounds per square inch.....	10,000	6,000	6,000
Moisture removed in heating, per cent.....		24	
Length of heating, minutes.....	15		Hot molds
Reduction of volume by pressing, per cent.....		48	40
Crushing strength, pounds per square inch.....			2,800
Quality of briquet:			
Fresh.....	C	C	C
Air-dried.....	A	B	A

^c Test 86 was made with a mixture of 50 per cent each of Pittsburgh slack and anthracite culm.

A sample of Pennsylvania anthracite culm and 3 per cent of cell pitch made strong briquets. However, better results were obtained by heating the mixture with a gas flame before pressing than when a steam jet was used.

With 10 per cent of sulphite liquor pressed cold, the briquet was very soft while fresh and had hardly strength enough to stand conveyance to storage. With 5 per cent of sulphite liquor heated over a gas flame before pressing, excellent briquets were obtained.

TESTS WITH PHILIPPINE LIGNITE.

The data obtained from laboratory tests of Philippine lignite (Pittsburgh No. 112^a) and various binders are given in the following tables:

Laboratory briquetting tests of Philippine lignite without binder.

	Test No.				
	L 97.	L 98.	L 117.	L 118.	L 119.
Size as used:					
Over $\frac{1}{2}$ inch, per cent.....	8.3	8.3	0.0	0.0	1.0
$\frac{1}{2}$ to $\frac{3}{4}$ inch, per cent.....	50.3	50.3	0.0	0.0	14.0
$\frac{3}{4}$ to 1 inch, per cent.....	24.8	24.8	0.0	0.0	30.0
1 to $\frac{3}{4}$ inch, per cent.....	9.6	9.6	0.0	0.0	27.7
Through $\frac{3}{4}$ inch, per cent.....	7.0	7.0	100.0	100.0	27.3
Briquetting temperature, °F.....	Hot.	Hot.	(^b)	(^b)	(^b)
Binder, amount used, per cent.....	None.	None.	None.	None.	None.
Method of heating used.....	S. J.	S. J.	S. J.	S. J.	S. J.
Weight of:					
Mixture used, grams.....	(^b)	180	200	150	150
Briquet, average, grams.....	(^b)	(^c)	(^c)	(^c)	(^c)
Condition of fuel, dried or undried.....	Undried.	Dried.	Dried.	Dried.	Undried.
Pressure used, pounds per square inch.....	14,000	14,000	14,000	30,000	30,000
Moisture in fuel, per cent.....	20.0	6.0	10.0	10.0	20.0
Moisture added in heating, per cent.....	(^b)	(^b)	(^b)	(^b)	(^b)
Moisture in briquet, per cent.....	(^b)	(^b)	(^b)	(^b)	(^b)
Length of heating, minutes.....			2	2	2
Reduction of volume by pressing, per cent.....	48	44.2	(^b)	(^b)	(^b)
Quality of briquet:					
Fresh.....	E	E	E	E	E
Air-dried.....	E	E	E	E	E

^a See table, p. 140.

^b Not determined.

^c None made.

Laboratory briquetting tests of Philippine lignite with water-gas pitch binder.

	Test No.					
	L 98.	L 98.	L 99.	L 100.	L 101.	L 102.
Size as used:						
Over $\frac{1}{2}$ inch, per cent.....	8.3	8.3	0.0	0.0	1.0	1.0
$\frac{1}{2}$ to $\frac{1}{4}$ inch, per cent.....	50.3	50.3	54.8	54.8	14.0	14.0
$\frac{1}{4}$ to $\frac{1}{8}$ inch, per cent.....	24.8	24.8	27.0	27.0	30.0	30.0
$\frac{1}{8}$ to $\frac{1}{16}$ inch, per cent.....	9.6	9.6	10.6	10.6	27.7	27.7
Through $\frac{1}{16}$ inch, per cent.....	7.0	7.0	7.6	7.6	27.3	27.3
Briquetting temperature, °F.....	163	136	131	147	142	145
Binder:						
Lot No.....	26	26	26	26	26	26
Amount used, per cent.....	4	6	8	4	6	8
Method of heating used.....	S. J.	S. J.	S. J.	S. J.	S. J.	S. J.
Weight of:						
Mixture used, grams.....	200	200	200	200	200	200
Briquet, average, grams.....	206	205	203	209	208	208
Condition of fuel, dried or undried.....	Undried.	Undried.	Undried.	Dried.	Dried.	Dried.
Pressure used, pounds per square inch.....	6,000	6,000	6,000	6,000	6,000	6,000
Moisture:						
In fuel, per cent.....	20.0	20.0	20.0	5.0	5.0	5.0
Added in heating, per cent.....	6.9	4.6	4.5	5.0	4.5	4.5
In briquet, per cent.....	26.9	24.6	24.5	10.0	9.5	9.5
Length of heating, minutes.....	2	2	2	2	2	2
Reduction of volume by pressing, per cent.....	43.2	29.7	40.7	29.4	43.0	41.0
Crushing strength, pounds per square inch.....	950	1,400	1,675	625	1,050	1,550
Quality of briquet:						
Fresh.....	C	B	A	D	B	A
Air-dried.....	B	A	A	C	A	A

Laboratory briquetting tests of Philippine lignite with wheat-flour binder.

	Test No.					
	L 144.	L 156.	L 115.	L 157.	L 184.	L 146.
Size as used:						
Over $\frac{1}{2}$ inch, per cent.....	0.0	0.0	1.0	0.0	0.0	0.0
$\frac{1}{2}$ to $\frac{1}{4}$ inch, per cent.....	14.1	14.1	14.0	14.1	14.1	14.1
$\frac{1}{4}$ to $\frac{1}{8}$ inch, per cent.....	30.6	30.6	30.0	30.6	30.6	30.6
$\frac{1}{8}$ to $\frac{1}{16}$ inch, per cent.....	28.0	28.0	27.7	28.0	28.0	28.0
Through $\frac{1}{16}$ inch, per cent.....	27.3	27.3	27.3	27.3	27.3	27.3
Briquetting temperature, °F.....	191	193	138	192	172	197
Binder:						
Lot No.....	31	31	31	31	31	31
Amount used, per cent.....	4	5	6	6	6	8
Method of heating used.....	S. J.	S. J.	S. J.	S. J.	S. J.	S. J.
Weight of:						
Mixture used, grams.....	200	200	200	200	200	200
Briquet, average, grams.....	210	211	210	210	210	209
Condition of fuel, dried or undried.....	Undried.	Undried.	Undried.	Undried.	Undried.	Undried.
Pressure used, pounds per square inch.....	6,000	6,000	6,000	6,000	6,000	6,000
Moisture:						
In fuel, per cent.....	20.0	20.0	20.0	20.0	20.0	20.0
Added in heating, per cent.....	7.5	7.0	8.0	7.0	6.0	6.5
In briquet, per cent.....	27.5	27.0	28.0	27.0	26.0	26.5
Length of heating, minutes.....	1 $\frac{1}{2}$	1	2	1	1	1
Reduction of volume by pressing, per cent.....	44.5	44.6	46.1	44.6	45.0	43.6
Crushing strength, pounds per square inch.....	1,550	1,400	1,550	1,425	1,200	1,975
Quality of briquet:						
Fresh.....	B	B	B	B	A	A
Air-dried.....	A	A	A	B	A	A

a Water cracks due to too much heating.

Laboratory briquetting tests of Philippine lignite with wheat-flour binder—Continued.

	Test No.		
	L 111.	L 112.	L 113.
Size as used:			
Over $\frac{1}{8}$ inch, per cent.....	1.0	1.0	1.0
$\frac{1}{8}$ to $\frac{1}{4}$ inch, per cent.....	14.0	14.0	14.0
$\frac{1}{4}$ to $\frac{3}{8}$ inch, per cent.....	30.0	30.0	30.0
$\frac{3}{8}$ to $\frac{1}{2}$ inch, per cent.....	27.7	27.7	27.7
Through $\frac{1}{2}$ inch, per cent.....	27.3	27.3	27.3
Briquetting temperature, °F.....	145	149	153
Binder:			
Lot No.....	3i	3i	3i
Amount used, per cent.....	4	6	8
Method of heating used.....	S. J.	S. J.	S. J.
Weight of:			
Mixture used, grams.....	200	200	200
Briquet, average, grams.....	204	205	204
Condition of fuel, dried or undried.....	Dried.	Dried.	Dried.
Pressure used, pounds per square inch.....	6,000	6,000	6,000
Moisture:			
In fuel, per cent.....	5.0	5.0	5.0
Added in heating, per cent.....	4.5	4.5	4.5
In briquet, per cent.....	9.5	9.5	9.5
Length of heating, minutes.....	1½	1½	1½
Reduction of volume by pressing, per cent.....	44.8	43.1	44.8
Crushing strength, pounds per square inch.....	0.0	0.0	0.0
Quality of briquet:			
Fresh.....	E	E	E
Air-dried.....	E	E	E

Laboratory briquetting tests of Philippine lignite with cornstarch binder.

	Test No.					
	L 138.	L 139.	L 103.	L 104.	L 105.	L 106.
Size as used:						
Over $\frac{1}{8}$ inch, per cent.....	1.0	1.0	1.0	1.0	1.0	1.0
$\frac{1}{8}$ to $\frac{1}{4}$ inch, per cent.....	14.0	14.0	14.0	14.0	14.0	14.0
$\frac{1}{4}$ to $\frac{3}{8}$ inch, per cent.....	30.0	30.0	30.0	30.0	30.0	30.0
$\frac{3}{8}$ to $\frac{1}{2}$ inch, per cent.....	27.7	27.7	27.7	27.7	27.7	27.7
Through $\frac{1}{2}$ inch, per cent.....	27.3	27.3	27.3	27.3	27.3	27.3
Briquetting temperature, °F.....	181	156	147	149	149	149
Binder:						
Lot No.....	35	35	35	35	35	35
Amount used, per cent.....	3	5	3	5	6	7
Method of heating used.....	S. J.	S. J.	S. J.	S. J.	S. J.	S. J.
Weight of:						
Mixture used, grams.....	200	200	200	200	200	200
Briquet, average, grams.....	210	205	203	204	202	201
Condition of fuel, dried or undried.....	Undried.	Undried.	Dried.	Dried.	Dried.	Dried.
Pressure used, pounds per square inch.....	6,000	6,000	6,000	6,000	6,000	6,000
Moisture:						
In fuel, per cent.....	20.0	20.0	5.0	5.0	5.0	5.0
Added in heating, per cent.....	7.0	3.5	4.5	4.5	3.5	2.5
In briquet, per cent.....	27.0	23.5	9.5	9.5	8.5	7.5
Length of heating, minutes.....	2	½	½	1½	1½	1½
Reduction of volume by pressing, per cent.....	45.3	44.6	38.5	37.0	38.5	35.5
Crushing strength, pounds per square inch.....	1,150	1,225	0.0	0.0	0.0	0.0
Quality of briquet:						
Fresh.....	D	C	D	D	D	D
Air-dried.....	B	B	D	a D	a D	a D

a Briquets were too dry.

Laboratory briquetting tests of Philippine lignite with cell-pitch binder.

	Test No.					
	L 140.	L 143.	L 142.	L 141.	L 107.	L 108.
Size as used:						
Over $\frac{1}{2}$ inch, per cent.....	1.0	1.0	1.0	1.0	1.0	1.0
$\frac{1}{2}$ to $\frac{3}{4}$ inch, per cent.....	14.0	14.0	14.0	14.0	14.0	14.0
$\frac{3}{4}$ to $\frac{1}{2}$ inch, per cent.....	30.0	30.0	30.0	30.0	30.0	30.0
$\frac{1}{2}$ to $\frac{1}{4}$ inch, per cent.....	27.7	27.7	27.7	27.7	27.7	27.7
Through $\frac{1}{4}$ inch, per cent.....	27.3	27.3	27.3	27.3	27.3	27.3
Briquetting temperature, °F.....	148	180	167	152	147	135
Binder:						
Lot No.....	27	27	27	27	27	27
Amount used, per cent.....	4	5	6	7	4	5
Method of heating used.....	S. J.	S. J.	S. J.	S. J.	S. J.	S. J.
Weight of:						
• Mixture used, grams.....	200	200	200	200	200	200
Briquet, average, grams.....	204	207	206	205	209	210
Condition of fuel, dried or undried.....	Undried.	Undried.	Undried.	Undried.	Dried.	Dried.
Pressure used, pounds per square inch.....	6,000	6,000	6,000	6,000	6,000	6,000
Moisture:						
In fuel, per cent.....	20.3	20.3	20.3	20.3	5.0	5.0
Added in heating, per cent.....	3.5	4.5	4.5	4.0	5.5	6.0
In briquet, per cent.....	23.8	24.8	24.8	24.3	10.5	11.0
Length of heating, minutes.....	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$
Reduction of volume by pressing, per cent.....	40.0	41.6	41.6	41.6	42.8	41.3
Crushing strength, pounds per square inch.....	800	900	925	950	475	450
Quality of briquet:						
Fresh.....	D	D	D	C	D	D
Air-dried.....	C	C	B	B	D	D

	Test No.			
	L 109.	L 145.	L 110.	L 147.
Size as used:				
Over $\frac{1}{2}$ inch, per cent.....	1.0	0.0	1.0	0.0
$\frac{1}{2}$ to $\frac{3}{4}$ inch, per cent.....	14.0	14.1	14.0	14.1
$\frac{3}{4}$ to $\frac{1}{2}$ inch, per cent.....	30.0	30.6	30.0	30.6
$\frac{1}{2}$ to $\frac{1}{4}$ inch, per cent.....	27.7	28.0	27.7	28.0
Through $\frac{1}{4}$ inch, per cent.....	27.3	27.3	27.3	27.3
Briquetting temperature, °F.....	131	176	138	196
Binder:				
Lot No.....	27	27	27	27
Amount used, per cent.....	6	6	7	6
Method of heating used.....	S. J.	S. J.	S. J.	S. J.
Weight of:				
Mixture used, grams.....	200	200	200	200
Briquet, average, grams.....	210	207	209	208
Condition of fuel, dried or undried.....	Dried.	Dried.	Dried.	Dried.
Pressure used, pounds per square inch.....	6,000	6,000	6,000	6,000
Moisture:				
In fuel, per cent.....	5.0	10.0	5.0	15.0
Added in heating, per cent.....	6.0	4.5	6.0	5.5
In briquet, per cent.....	11.0	14.5	11.0	20.5
Length of heating, minutes.....	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$
Reduction of volume by pressing, per cent.....	42.9	41.5	42.6	44.6
Crushing strength, pounds per square inch.....	575	775	775	950
Quality of briquet:				
Fresh.....	D	C	C	C
Air-dried.....	C	B	C	B

Laboratory briquetting tests of Philippine lignite with sulphite-liquor binder.

	Test No.					
	L 148.	L 149.	L 150.	L 151.	L 116.	L 152.
Size as used:						
Over $\frac{1}{4}$ inch, per cent.....	0.0	0.0	0.0	0.0	1.0	1.0
$\frac{1}{4}$ to $\frac{1}{2}$ inch, per cent.....	14.1	14.1	14.1	14.1	14.1	14.1
$\frac{1}{2}$ to $\frac{3}{4}$ inch, per cent.....	30.6	30.6	30.6	30.6	30.0	30.6
$\frac{3}{4}$ to 1 inch, per cent.....	28.0	28.0	28.0	28.0	27.7	27.7
Through 1 inch, per cent.....	27.3	27.3	27.3	27.3	27.3	27.3
Briquetting temperature, °F.....	211	212	210	211	218	208
Binder:						
Lot No.....	34	34	34	34	34	34
Amount used, per cent.....	4	5	6	7	4	4
Method of heating used.....	S. J.	S. J.	S. J.	S. J.	S. J.	S. J.
Weight of:						
Mixture used, grams.....	200	200	200	200	200	200
Briquet, average, grams.....	207	208	208	208	216	206
Condition of fuel, dried or undried.....	Undried.	Undried.	Undried.	Undried.	Dried.	Dried.
Pressure used, pounds per square inch.....	6,000	6,000	6,000	6,000	6,000	6,000
Moisture:						
In fuel, per cent.....	20.3	20.3	20.3	20.3	5.0	7.1
Added in heating, per cent.....	5.5	6.0	5.0	5.5	9.0	4.0
In briquet, per cent.....	25.8	26.3	25.3	25.8	14.0	11.1
Length of heating, minutes.....	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$
Reduction of volume by pressing, per cent.....	38.4	41.6	40.0	40.0	42.0	35.0
Crushing strength, pounds per square inch.....	725	775	600	625	450	500
Quality of briquet:						
Fresh.....	D	D	D	C	D	D
Air-dried.....	C	C	C	C	C	C

	Test No.		
	L 153.	L 154.	L 155.
Size as used:			
Over $\frac{1}{4}$ inch, per cent.....	1.0	1.0	1.0
$\frac{1}{4}$ to $\frac{1}{2}$ inch, per cent.....	14.0	14.0	14.0
$\frac{1}{2}$ to $\frac{3}{4}$ inch, per cent.....	30.0	30.0	30.0
$\frac{3}{4}$ to 1 inch, per cent.....	27.7	27.7	27.7
Through 1 inch, per cent.....	27.3	27.3	27.3
Briquetting temperature, °F.....	209	211	210
Binder:			
Lot No.....	34	34	34
Amount used, per cent.....	5	8	7
Method of heating used.....	S. J.	S. J.	S. J.
Weight of:			
Mixture used, grams.....	200	200	200
Briquet, average, grams.....	207	209	209
Condition of fuel, dried or undried.....	Dried.	Dried.	Dried.
Pressure used, pounds per square inch.....	6,000	6,000	6,000
Moisture:			
In fuel, per cent.....	7.1	7.1	7.1
Added in heating, per cent.....	4.5	5.5	5.0
In briquet, per cent.....	11.6	12.6	12.1
Length of heating, minutes.....	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$
Reduction of volume by pressing, per cent.....	36.5	40.0	43.1
Crushing strength, pounds per square inch.....	500	725	700
Quality of briquet:			
Fresh.....	D	D	D
Air-dried.....	C	C	D

All tests without binder failed to produce briquets that could be taken from the mold without breaking.

With 6 per cent of water gas pitch binder fairly good briquets were made from this lignite either in a dried or undried condition; 8 per cent of the water gas pitch made excellent briquets when the fuel was dried; 4 per cent of the water-gas pitch made a fair briquet of the undried fuel.

As low as 4 per cent of wheat flour made excellent briquets of the undried fuel, but even 8 per cent was not sufficient to briquet the dried fuel containing 5 per cent of moisture. The reason may be found in the fact that to obtain the adhesive power of the flour it must absorb a large quantity of water and if the coal is too dry it absorbs the water from the steam used for heating, and the flour is not thoroughly glutinized.

The undried fuel and 6 per cent of cell pitch made satisfactory briquets although they required careful handling when fresh from the press. Those made with 8 per cent were much stronger when fresh, but when cold differed very little from the 6 per cent briquets. Comparison of tests Nos. 109 and 147 shows that briquets in which 15 per cent of moisture was present were much stronger than those with 5 per cent of moisture.

Sulphite liquor did not make a satisfactory binder either with the dried or undried fuel.

Five per cent of cornstarch with the undried fuel was not sufficient to produce a briquet that was strong when fresh. With the fuel dried to contain only 5 per cent of moisture, 7 per cent of cornstarch did not give satisfactory briquets, again illustrating the important part that the proper proportion of moisture plays in briquetting lignite.

The source and analysis of this fuel sample are to be found under "Details of Briquetting Tests."

TESTS WITH PITTSBURGH BITUMINOUS SLACK.

The data obtained from laboratory tests of Pittsburgh bituminous slack and various binders are given in the following tables:

Laboratory briquetting tests of Pittsburgh slack with water-gas pitch binder.

	Test No.		
	L 61.	L 68.	L 69.
Briquetting temperature, °F.	158	151	194
Binder:			
Lot No.	26	26	26
Amount used, per cent.	6	6	8
Method of heating used.	S. J.	S. J.	S. J.
Weight of:			
Mixture used, grams.	200	200	200
Briquet, average, grams.		198	198
Condition of fuel, dried or undried.	Undried.	Undried.	Undried.
Pressure used, pounds per square inch.	4,000	4,000	4,000
Crushing strength, pounds per square inch.	2,220	2,100	2,800
Quality of briquet:			
Fresh.	A	A	A
Air-dried.	A	A	A

Laboratory briquetting tests of Pittsburgh slack with cornstarch binder.

	Test No. L 70.
Briquetting temperature, °F.....	203
Binder:	
Lot No.....	25
Amount used, per cent.....	30
Method of heating used.....	S. J.
Weight of:	
Mixture used, grams.....	200
Briquet, average, grams.....	213
Condition of fuel, dried or undried.....	Undried.
Pressure used, pounds per square inch.....	4,000
Moisture added in heating, per cent.....	6.5
Crushing strength, pounds per square inch.....	2,100
Quality of briquet:	
Fresh.....	A
Air-dried.....	A

Laboratory briquetting tests of Pittsburgh slack with hardwood-tar pitch binder.

	Test No.				
	L 60.	L 59.	L 65.	L 62.	L 66.
Briquetting temperature, °F.....	158	158	167	158	167
Binder:					
Lot No.....	29	29	29	29	29
Amount used, per cent.....	a 4	a 6	b 4	b 6	b 8
Method of heating used.....	S. J.	S. J.	S. J.	S. J.	S. J.
Weight of:					
Mixture used, grams.....	200	200	200	200	200
Briquet, average, grams.....	200	200	200	200	200
Condition of fuel, dried or undried.....	Undried.	Undried.	Undried.	Undried.	Undried.
Pressure used, pounds per square inch.....	4,000	4,000	4,000	4,000	4,000
Crushing strength, pounds per square inch.....	1,750	1,600	1,960	2,275	2,100
Quality of briquet:					
Fresh.....	A	A	A	A	A
Air-dried.....	A	A	A	A	A

a Binder mixed cold.

b Binder used in melted form.

Laboratory briquetting tests of Pittsburgh slack with cell-pitch binder.

	Test No.			
	L 92.	L 88.	L 87A.	L 87B.
Briquetting temperature, °F.....	151	156	194	194
Binder:				
Lot No.....	27	27	27	27
Amount used, per cent.....	2	4	5	5
Method of heating used.....	S. J.	S. J.	S. J.	S. J.
Weight of:				
Mixture used, grams.....	200	200	200	200
Briquet, average, grams.....	205	206	204	204
Condition of fuel, dried or undried.....	Undried.	Undried.	Undried.	Undried.
Pressure used, pounds per square inch.....	6,000	6,000	6,000	6,000
Moisture added in heating, per cent.....	4.5	4.5		
Length of heating, minutes.....	½	½		
Condition of mold.....	Hot.	Hot.	Hot.	Hot.
Reduction of volume by pressing, per cent.....	37	41	39	39
Crushing strength, pounds per square inch.....	1,800	2,350		3,300
Quality of briquet:				
Fresh.....	B	B	B	B
Air-dried.....	A	A	A	A

Laboratory briquetting tests of Pittsburgh slack with cell-pitch binder—Continued.

	Test No.			
	L 89.	L 90.	L 91.	L 93.
Briquetting temperature, °F.....	158	144	158	143
Binder:				
Lot No.....	27	27	27	27
Amount used, per cent.....	6	8	8	10
Method of heating used.....	S. J.	S. J.	S. J.	S. J.
Weight of:				
Mixture used, grams.....	200	200	205	200
Briquet, average, grams.....	206	204	204	205
Condition of fuel, dried or undried.....	Undried.	Undried.	Undried.	Undried.
Pressure used, pounds per square inch.....	6,000	6,000	6,000	6,000
Moisture added in heating, per cent.....	4.5	3.0	2.5	4.0
Length of heating, minutes.....	½	½	½	½
Condition of mold.....	Hot and cold.	Hot and cold.	Cold.	Cold.
Reduction of volume by pressing, per cent.....	37	36	36	39
Crushing strength, pounds per square inch.....	3,800		3,800	3,700
Quality of briquet:		3,400		
Fresh.....	A	A	A	A
Air-dried.....	A	A	A	A

* Material stuck to plunger, showing excess of binder.

Laboratory briquetting tests of Pittsburgh slack with sulphite-liquor binder.

	Test No.					
	L 72.	L 73.	L 74A.	L 74B.	L 75A.	L 75B.
Briquetting temperature, °F.....	66	180	199	199	203	203
Binder:						
Lot No.....	34	34	34	34	34	34
Amount used, per cent.....	3	3	3	3	4	4
Method of heating used.....	Cold.	S. J.	S. J.	S. J.	S. J.	S. J.
Weight of mixture used, grams.....	200	200	200	200	200	200
Condition of fuel, dried or undried.....	Undried.	Undried.	Undried.	Undried.	Undried.	Undried.
Pressure used, pounds per square inch.....	4,000	4,000	4,000	4,000	4,000	4,000
Length of heating, minutes.....	Cold.	1½	1	1	1	1
Reduction of volume by pressing, per cent.....	33	29	35	34	33	33
Crushing strength, pounds per square inch.....	1,700		1,400	1,500	1,100	1,800
Quality of briquet:						
Fresh.....	C	C	B	B	C	B
Air-dried.....	A	B	A	A	B	A

	Test No.				
	L 76.	L 76B.	L 76C.	L 76D.	L 232.
Briquetting temperature, °F.....	Cold.	171	203	257	194
Binder:					
Lot No.....	34	34	34	34	32
Amount used, per cent.....	10	10	10	10	4
Method of heating used.....	Cold.	Hot mold only.	S. J.	Gas flame.	S. J.
Weight of:					
Mixture used, grams.....	200	200	200	200	200
Briquet, average, grams.....	200	200			206
Condition of fuel, dried or undried.....	Undried.	Undried.	Undried.	Undried.	Undried.
Pressure used, pounds per square inch.....	4,000	4,000	4,000	4,000	4,000
Moisture added in heating, per cent.....					4.0
Length of heating, minutes.....	Cold.	Hot.	½		½
Reduction of volume by pressing, per cent.....	36	36	36	40	44
Crushing strength, pounds per square inch.....	3,500	3,900			1,625
Quality of briquet:					
Fresh.....	A	A	C	D	A
Air-dried.....	A	A	B+	C	A

Laboratory briquetting tests of Pittsburg slack with sulphite-liquor binder—Continued.

	Test No.			
	L 233.	L 234.	L 235.	L 236.
Briquetting temperature, °F.....	194	196	189	192
Binder:				
Lot No.....	32	32	32	32
Amount used, per cent.....	6	8	10	12
Method of heating used.....	S. J.	S. J.	S. J.	S. J.
Weight of:				
Mixture used, grams.....	200	200	200	200
Briquet, average, grams.....	205	206	206	206
Condition of fuel, dried or undried.....	Undried.	Undried.	Undried.	Undried.
Pressure used, pounds per square inch.....	4,000	4,000	4,000	4,000
Moisture added in heating, per cent.....	3.5	3.5	4.0	3.5
Length of heating, minutes.....	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$
Reduction of volume by pressing, per cent.....	44	44	43	47
Crushing strength, pounds per square inch.....	1,875	2,300	2,725	3,000
Quality of briquet:				
Fresh.....	A	A	A	A
Air-dried.....	A	A	A	A

Laboratory briquetting tests of Pittsburgh slack with water-gas pitch and cell-pitch binders mixed.

	Test No. L 94.
Briquetting temperature, °F.....	194
Binder:	
Lot No.—	
26.....	W. G. P.
27.....	C. F.
Amount used, per cent.....	3 of each.
Method of heating used.....	S. J.
Weight of:	
Mixture used, grams.....	200
Briquet, average, grams.....	206
Condition of fuel, dried or undried.....	Undried.
Pressure used, pounds per square inch.....	6,000
Moisture added in heating, per cent.....	5.0
Length of heating, minutes.....	$\frac{1}{2}$
Reduction of volume by pressing, per cent.....	38
Quality of briquet:	
Fresh.....	a C
Air-dried.....	B

a Mixture was too wet.

Pittsburgh slack was easily briquetted with 6 per cent of water-gas pitch. All briquets made of this fuel and binder were of excellent quality.

A test (L 70) with cornstarch and bituminous slack showed that 3 per cent of this binder is sufficient to make excellent briquets of this fuel.

Hardwood-tar pitch was tested with Pittsburgh slack. This binder is soft enough to be cut with the finger nail at ordinary temperatures. For this reason it was with difficulty incorporated with the fuel when cold. Better mixtures were obtained by melting the binder and mixing it with the fuel. In test L 60 the binder and the fuel were mixed by stirring in a mortar. Tests L 65, L 62, and L 66 were made by mixing the binder in a melted condition. This binder made briquets of satisfactory strength when as low as 4 per cent was

used, but for commercial purposes 5 to 6 per cent would be required to give the briquet sufficient strength and hardness to withstand transporting into storage. A jet of steam is a satisfactory means of heating the mixture of fuel and molten pitch. This binding material has a strong characteristic creosote odor, and should have good disinfecting properties. The presence of this odor might prevent the use of this binder for household briquets.

Cell pitch was also tested as a binder for Pittsburgh slack—2, 4, 5, 6, 8, and 10 per cent being used. It is interesting to note that only 2 per cent of this binder produced a briquet having a crushing strength of 1,800 pounds per square inch when cold and air-dried. A 6 per cent water gas pitch briquet has a crushing strength of only about 2,100 pounds per square inch. The higher percentages of this binder made stronger briquets. It should be said, however, that when fresh the briquets made with 2, 4, and 5 per cent of this binder were slightly soft. If the mixture is heated with superheated steam, the moisture being thus kept low, there should be no difficulty in producing satisfactory briquets with a low percentage of this binder. The great drawback with this binder is that it is soluble in water and easily leached by a rainstorm, allowing the fuel particles to fall to pieces.

Experiments were made with Pittsburgh slack and 3 to 12 per cent of sulphite-liquor binder. Although 3 per cent of this binder is sufficient to produce briquets that are strong when cold and air-dried, they are fragile when fresh and would not stand commercial handling from the press to storage. It is necessary to use a larger proportion of this binder to make briquets of sufficient strength when fresh than is necessary for a dried and cool briquet. It is therefore desirable that the sulphite liquor be used in as concentrated a form as possible.

A test (L 94) was made with Pittsburgh slack and 3 per cent each of water-gas pitch and cell pitch to determine the waterproofing qualities of the former in connection with the strong qualities of the latter binding material. When fresh, the resultant briquets were very weak and required careful handling, but after having been dried and cooled they were much stronger and appeared to possess satisfactory water-resisting qualities.

TESTS WITH TEXAS LIGNITE.

The data obtained from laboratory tests of Texas lignite (Pittsburgh No. 9 ^a) and various binders are given in the following tables:

^a See table, p. 40.

Laboratory briquetting tests of Texas lignite with water-gas pitch binder.

	Test No.				
	L 187.	L 188.	L 189.	L 190.	L 191.
Size as used:					
Over $\frac{1}{2}$ inch, per cent.....	7.2	7.2	7.2	7.2	7.2
$\frac{1}{2}$ to $\frac{3}{4}$ inch, per cent.....	30.6	30.6	30.6	30.6	30.6
$\frac{3}{4}$ to $\frac{1}{2}$ inch, per cent.....	30.2	30.2	30.2	30.2	30.2
$\frac{1}{2}$ to $\frac{1}{4}$ inch, per cent.....	15.9	15.9	15.9	15.9	15.9
Through $\frac{1}{4}$ inch, per cent.....	16.1	16.1	16.1	16.1	16.1
Briquetting temperature, °F.....	196	196	192	196	189
Binder:					
Lot No.....	26	26	26	26	26
Amount used, per cent.....	4	5	6	7	8
Method of heating used.....	S. J.	S. J.	S. J.	S. J.	S. J.
Weight of:					
Mixture used, grams.....	210	211	210	211	210
Briquet, average, grams.....	206	206	206	208	206
Condition of fuel, dried or undried.....	Undried.	Undried.	Undried.	Undried.	Undried.
Pressure used, pounds per square inch.....	4,000	4,000	4,000	4,000	4,000
Moisture:					
In fuel, per cent.....	18.2	18.2	18.2	18.2	18.2
Added in heating, per cent.....	5.0	5.5	5.0	5.5	5.0
In briquet, per cent.....	23.2	23.7	23.2	23.7	23.2
Length of heating, minutes.....	$\frac{1}{2}$ and $\frac{1}{4}$	$\frac{1}{2}$ and $\frac{1}{4}$	$\frac{1}{2}$ and $\frac{1}{4}$	$\frac{1}{2}$ and $\frac{1}{4}$	$\frac{1}{2}$ and $\frac{1}{4}$
Reduction of volume by pressing, per cent.....	42	43	43.8	44.8	42.8
Crushing strength, pounds per square inch.....	400	650	800	675	950
Quality of briquet:					
Fresh.....	C	C	B	B	A
Air-dried.....	C	C	B	B	A

Laboratory briquetting tests of Texas lignite with flour binder.

	Test No.						
	L 192.	L 193.	L 194.	L 195.	L 196.	L 197.	L 198.
Size as used:							
Over $\frac{1}{2}$ inch, per cent.....	7.2	7.2	7.2	7.2	7.2	7.2	7.2
$\frac{1}{2}$ to $\frac{3}{4}$ inch, per cent.....	30.6	30.6	30.6	30.6	30.6	30.6	30.6
$\frac{3}{4}$ to $\frac{1}{2}$ inch, per cent.....	30.2	30.2	30.2	30.2	30.2	30.2	30.2
$\frac{1}{2}$ to $\frac{1}{4}$ inch, per cent.....	15.9	15.9	15.9	15.9	15.9	15.9	15.9
Through $\frac{1}{4}$ inch, per cent.....	16.1	16.1	16.1	16.1	16.1	16.1	16.1
Briquetting temperature, °F.....	196	194	196	196	196	196	196
Binder:							
Lot No.....	31	31	31	31	31	31	31
Amount used, per cent.....	2	3	4	5	6	7	8
Method of heating used.....	S. J.	S. J.	S. J.	S. J.	S. J.	S. J.	S. J.
Weight of:							
Mixture used, grams.....	212	213	213	212	212	212	212
Briquet, average, grams.....	209	209	210	209	209	208	209
Condition of fuel, dried or undried.....	Undried.	Undried.	Undried.	Undried.	Undried.	Undried.	Undried.
Pressure used, pounds per square inch.....	4,000	4,000	4,000	4,000	4,000	4,000	4,000
Moisture:							
In fuel, per cent.....	18.2	18.2	18.2	18.2	18.2	18.2	18.2
Added in heating, per cent.....	6.0	6.5	6.5	6.0	6.0	6.0	6.0
In briquet, per cent.....	24.2	24.7	24.7	24.2	24.2	24.2	24.2
Length of heating, minutes.....	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Reduction of volume by pressing, per cent.....	46	46	46	46	46	46	46
Crushing strength, pounds per square inch.....	750	900	1,200	1,325	1,375	1,450	1,475
Quality of briquet:							
Fresh.....	B	B	A	A	A	A	A
Air-dried.....	B	B	A	A	A	A	A

Laboratory briquetting tests of Texas lignite with cell-pitch binder.

	Test No. L 123.
Briquetting temperature, °F.....	162
Binder:	
Lot No.....	27
Amount used, per cent.....	8
Method of heating used.....	S. J.
Weight of:	
Mixture used, grams.....	210
Briquet, average, grams.....	207
Condition of fuel, dried or undried.....	Undried.
Pressure used, pounds per square inch.....	6,000
Moisture:	
In fuel, per cent.....	18.2
Added in heating, per cent.....	5.0
In briquet, per cent.....	23.2
Length of heating, minutes.....	$\frac{1}{2}$
Crushing strength, pounds per square inch.....	1,250
Quality of briquet:	
Fresh.....	A
Air-dried.....	A

Laboratory briquetting tests of Texas lignite with sulphite-liquor binder.

	Test No.						
	L 211.	L 212.	L 213.	L 214.	L 215.	L 216.	L 217.
Size as used:							
Over $\frac{1}{2}$ inch, per cent.....	7.2	7.2	7.2	7.2	7.2	7.2	7.2
$\frac{1}{2}$ to $\frac{3}{4}$ inch, per cent.....	30.6	30.6	30.6	30.6	30.6	30.6	30.6
$\frac{3}{4}$ to 1 inch, per cent.....	30.2	30.2	30.2	30.2	30.2	30.2	30.2
1 to $1\frac{1}{4}$ inch, per cent.....	15.9	15.9	15.9	15.9	15.9	15.9	15.9
Through $1\frac{1}{4}$ inch, per cent.....	16.1	16.1	16.1	16.1	16.1	16.1	16.1
Briquetting temperature, °F.....	192	199	196	194	199	194	181
Binder:							
Lot No.....	34	34	34	34	34	34	34
Amount used, per cent.....	3	4	5	6	7	8	9
Method of heating used.....	S. J.	S. J.	S. J.	S. J.	S. J.	S. J.	S. J.
Weight of:							
Mixture used, grams.....	208	210	210	210	210	210	209
Briquet, average, grams.....	206	208	207	207	207	206	206
Condition of fuel, dried or undried.....	Undried.	Undried.	Undried.	Undried.	Undried.	Undried.	Undried.
Pressure used, pounds per square inch.....	4,000	4,000	4,000	4,000	4,000	4,000	4,000
Moisture:							
In fuel, per cent.....	18.2	18.2	18.2	18.2	18.2	18.2	18.2
Added in heating, per cent.....	4.0	5.0	5.0	5.0	5.0	5.0	4.5
In briquet, per cent.....	22.2	23.2	23.2	23.2	23.2	23.2	22.7
Length of heating, minutes.....	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$	$\frac{1}{2}$
Reduction of volume by pressing, per cent.....	46	46	45.5	45.5	47	47	47
Crushing strength, pounds per square inch.....	300	300	300	526	525	625	725
Quality of briquet:							
Fresh.....	D	C	C	C	C	C	C
Air-dried.....	D	C	C	C	C	C	B

This lignite was not successfully briquetted without binder. Tests with this fuel and water-gas pitch showed that not less than 6 per cent of binder would give a fair briquet, and that even 8 per cent did not furnish what might be called an excellent briquet. All tests were made on the undried fuel. Three per cent of flour made a good briquet, and 4 per cent produced an excellent briquet in both strength and appearance, although water would disintegrate it in a few hours. This fuel was made into excellent, although not waterproof, briquets

with 8 per cent of cell pitch. Nine per cent of sulphite liquor did not make entirely satisfactory briquets.

The source from which this sample was obtained and its chemical analysis are to be found under "Details of Briquetting Tests."

TESTS WITH UTAH SUBBITUMINOUS COAL.

The data obtained from laboratory tests of a sample of Utah subbituminous coal (Pittsburgh No. 36^a) and various binders are given in the following tables:

Laboratory briquetting tests of Utah subbituminous coal with water-gas pitch binder.

	Test No.	
	L 137.	L 132.
Briquetting temperature, °F.....	167	163
Binder:		
Lot No.....	26	26
Amount used, per cent.....	3	7
Method of heating used.....	S. J.	S. J.
Weight of:		
Mixture used, grams.....	200	200
Briquet, average, grams.....	207	206
Condition of fuel, dried or undried.....	Undried.	Undried.
Pressure used, pounds per square inch.....	6,000	6,000
Moisture:		
In fuel, per cent.....	10.8	10.8
Added in heating, per cent.....	4.5	4.0
In briquet, per cent.....	15.3	14.8
Length of heating, minutes.....	3	3
Reduction of volume by pressing, per cent.....	40	42
Crushing strength, pounds per square inch.....	200	1,050
Quality of briquet:		
Fresh.....	D	B
Air-dried.....	D	A

Laboratory briquetting tests of Utah subbituminous coal with cell-pitch binder.

	Test No.	
	L 134.	L 133.
Briquetting temperature, °F.....	164	159
Binder:		
Lot No.....	27	27
Amount used, per cent.....	4	7
Method of heating used.....	S. J.	S. J.
Weight of:		
Mixture used, grams.....	200	200
Briquet, average, grams.....	207	208
Condition of fuel, dried or undried.....	Undried.	Undried.
Pressure used, pounds per square inch.....	6,000	6,000
Moisture:		
In fuel, per cent.....	10.8	10.8
Added in heating, per cent.....	4.5	4.5
In briquet, per cent.....	15.3	15.3
Length of heating, minutes.....	3	3
Reduction of volume by pressing, per cent.....	40	42
Crushing strength, pounds per square inch.....	1,150	1,600
Quality of briquet:		
Fresh.....	C	B
Air-dried.....	A	A

^a See table, p. 40.

Laboratory briquetting tests of Utah subbituminous coal with a mixture of water-gas pitch and cell-pitch binders and of water-gas pitch and flour binders.

	Test No.	
	L 136.	L 135.
Briquetting temperature, °F.....	180	150
Binder:		
Lot No.	26 W. G. P.	26 W. G. P.
Amount used, per cent.	27 C. P.	31 flour.
Method of heating used.....	3 W. G. P.	3 W. G. P.
Weight of:	4 C. P.	3 flour.
Mixture used, grams.....	S. J.	S. J.
Briquet, average, grams.....	200	200
Condition of fuel, dried or undried.....	207	206
Pressure used, pounds per square inch.....	Undried.	Undried.
Moisture:	6,000	6,000
In fuel, per cent.....	10.8	10.8
Added in heating, per cent.....	4.5	4.0
In briquet, per cent.....	15.3	14.8
Length of heating, minutes.....	40	40
Reduction of volume by pressing, per cent.....	40	40
Crushing strength, pounds per square inch.....	1,060	1,450
Quality of briquet:		
Fresh.....	B	B
Air-dried.....	A	A

This fuel required 7 per cent of water-gas pitch to produce satisfactory briquets. Four per cent of cell pitch was sufficient to make strong briquets when they were cold and air-dried, but when fresh from the press they were rather weak. Probably 5 to 6 per cent of cell pitch would be required to make a commercial briquet.

Two tests, L 135 and L 136, were made to determine whether an addition of water-gas pitch would make cell-pitch or flour briquets sufficiently waterproof for practical use and storage out of doors. After immersion in water for 10 minutes (test L 136) the cell pitch began to leach out, whereas no change was noticed in the flour and water gas pitch briquet (test L 135). After a three-day immersion both briquets were strong enough to be handled when wet. The flour-and-pitch briquet was in better condition than the cell-pitch and water gas pitch briquet. The water in which the briquets were immersed was colored brown by the dissolved cell pitch. There was no evidence of the flour having been leached out.

TESTS WITH WASHINGTON SUBBITUMINOUS COAL.

The data obtained from laboratory tests of Washington subbituminous coal (Pittsburgh No. 35 ^a) and various binders are given in the following tables:

Laboratory briquetting tests of Washington subbituminous coal with water-gas pitch binder.

	Test No.	
	L 124.	L 125.
Briquetting temperature, °F	155	152
Binder:		
Lot No.	26	26
Amount used, per cent.	6	8
Method of heating used	S. J.	S. J.
Weight of:		
Mixture used, grams.	200	200
Briquet, average, grams.	205	205
Condition of fuel, dried or undried	Undried.	Undried.
Pressure used, pounds per square inch	6,000	6,000
Moisture:		
In fuel, per cent.	13.8	13.8
Added in heating, per cent.	4.5	3.5
In briquet, per cent.	18.3	17.3
Length of heating, minutes.	1	1
Reduction of volume by pressing, per cent.	42	42
Crushing strength, pounds per square inch	1,200	1,825
Quality of briquet:		
Fresh	A	A
Air-dried	A	A

Laboratory briquetting tests of Washington subbituminous coal with wheat-flour binder.

	Test No.	
	L 128.	L 129.
Briquetting temperature, °F	173	180
Binder:		
Lot No.	31	31
Amount used, per cent.	5	7
Method of heating used	S. J.	S. J.
Weight of:		
Mixture used, grams.	200	200
Briquet, average, grams.	210	209
Condition of fuel, dried or undried	Undried.	Undried.
Pressure used, pounds per square inch	6,000	6,000
Moisture:		
In fuel, per cent.	18.3	18.3
Added in heating, per cent.	6.0	6.0
In briquet, per cent.	24.3	24.3
Length of heating, minutes.	1	1
Reduction of volume by pressing, per cent.	41	40
Crushing strength, pounds per square inch	1,500	1,750
Quality of briquet:		
Fresh	A	A
Air-dried	A	A

^a See table, p. 41.

Laboratory briquetting tests of Washington subbituminous coal with cornstarch binder.

	Test No.	
	L 130.	L 131.
Briquetting temperature, °F.....	184	183
Binder:		
Lot No.....	35	35
Amount used, per cent.....	5	7
Method of heating used.....	S. J.	S. J.
Weight of:		
Mixture used, grams.....	200	200
Briquet, average, grams.....	208	208
Condition of fuel, dried or undried.....	Undried.	Undried.
Pressure used, pounds per square inch.....	6,000	6,000
Moisture:		
In fuel, per cent.....	18.3	18.3
Added in heating, per cent.....	6.0	6.0
In briquet, per cent.....	24.3	24.3
Length of heating, minutes.....	1	1
Reduction of volume by pressing, per cent.....	42	42
Crushing strength, pounds per square inch.....	1,850	2,100
Quality of briquet:		
Fresh.....	A	A
Air-dried.....	A	A

Laboratory briquetting tests of Washington subbituminous coal with cell-pitch binder.

	Test No.	
	L 126.	L 127.
Briquetting temperature, °F.....	161	161
Binder:		
Lot No.....	27	27
Amount used, per cent.....	6	5
Method of heating used.....	S. J.	S. J.
Weight of:		
Mixture used, grams.....	200	200
Briquet, average, grams.....	206	206
Condition of fuel, dried or undried.....	Undried.	Undried.
Pressure used, pounds per square inch.....	6,000	6,000
Moisture:		
In fuel, per cent.....	18.3	18.3
Added in heating, per cent.....	4.0	4.0
In briquet, per cent.....	22.3	22.3
Length of heating, minutes.....	1	1
Reduction of volume by pressing, per cent.....	42	42
Crushing strength, pounds per square inch.....	1,000	1,400
Quality of briquet:		
Fresh.....	B	A
Air-dried.....	A	A

The fuel made excellent briquets with each of the following binders: Six per cent of water-gas pitch; 5 per cent of wheat flour; 5 per cent of cornstarch; and 6 per cent of cell pitch. More than the above proportions of binder should not be required to make satisfactory briquets. In the case of flour and cornstarch the compressive strength of the 5 per cent briquets indicates that probably 3 per cent of either of these binders would be sufficient to produce a briquet of satisfactory strength for all purposes. The flour, cornstarch, and cell pitch would crumble in water and therefore would have to be stored under cover.

SUMMARY OF LABORATORY TESTS OF BINDERS.

In the following table are recorded the results of the laboratory tests of various binders and fuels as detailed above:

Summary of results of laboratory briquetting tests of various fuels and binders.

Fuel.	Binder.	Smallest per cent that made satisfactory briquet.	Crushing strength of briquet made with minimum of binder.	Crushing strength of briquet made with 6 per cent of binder.	Weather-resisting qualities of 6 per cent briquet.
Pittsburgh slack.....	Water-gas pitch ..	a 6.0	2,220	2,220	Excellent.
	Cornstarch	3.0	2,100	N. D.	Fair.
	Hardwood-tarpitch ..	4.0	1,960	2,275	Good.
	Cell pitch	2.0	1,800	3,800	Poor.
	Sulphite liquor.....	3.0	1,400	1,875	Very poor.
Texas lignite (P-9).....	Water-gas pitch ..	6.0	800	800	Good.
	Wheat flour.....	3.0	900	1,375	Fair.
	Cell pitch	a 8.0	1,250		Poor.
	Sulphite liquor.....	b 9.0	725	525	Very poor.
Pennsylvania anthracite culm.....	Cell pitch	3.0	2,700		Poor.
	Sulphite liquor.....	5.0	2,500		Very poor.
North Dakota lignite (P-15).....	Water-gas pitch ..	b 8.0	650	300	Fair.
	Wheat flour.....	3.0	750	1,125	Do.
	Cell pitch	8.0	925	550	Poor.
North Dakota lignite (P-17).....	Water-gas pitch ..	b 8.0	350	400	Fair.
	Wheat flour.....	5.0	1,125	1,200	Do.
	Sulphite liquor.....	b 9.0	300		Very poor.
Philippine lignite.....	Water-gas pitch ..	6.0	1,400	1,400	Good.
	Wheat flour.....	4.0	1,550	1,550	Fair.
	Cornstarch	b 5.0	1,225		Do.
	Cell pitch	6.0	925	925	Poor.
	Sulphite liquor.....	b 8.0	725	600	Very poor.
Utah subbituminous (P-36).....	Water-gas pitch ..	7.0	1,050		Good.
	Cell pitch	4.0	1,150		Poor.
Washington subbituminous (P-36).....	Water-gas pitch ..	6.0	1,200	1,200	Good.
	Wheat flour.....	a 5.0	1,500		Fair.
	Cornstarch	a 5.0	1,850		Do.
	Cell pitch	6.0	1,000	1,000	Poor.

^a Lowest percentage used in tests, but not necessarily the lowest that would furnish a satisfactory briquet.

^b This percentage was not sufficient to make entirely satisfactory briquets, but was the highest used in the tests.

CONCLUSIONS FROM LABORATORY TESTS.

Following are presented conclusions derived from the tests:

1. Different binders require different methods of heating to obtain the best results.
2. The various classes of fuels require different methods of treatment to produce the best briquets.
3. Lignites, to produce the strongest briquets, generally require drying before being mixed with binder, but for the best results some moisture must be allowed to remain in the fuel.
4. Six per cent of water-gas pitch made satisfactory briquets from Pittsburgh slack, Texas lignite, Philippine lignite, and Washington subbituminous coal; 7 per cent was sufficient to satisfactorily briquet Utah subbituminous coal.
5. Three per cent wheat flour made satisfactory briquets from samples of Texas and North Dakota lignite;^a 4 per cent made satis-

^a One sample of North Dakota lignite required 5 per cent.

factory briquets from undried Philippine lignite; and 5 per cent was more than sufficient to make satisfactory briquets from Washington subbituminous coal.

6. Cornstarch gave practically the same results as wheat flour, 3 to 5 per cent being required to make a satisfactory briquet from the various fuels.

7. Four per cent of hardwood-tar pitch made strong briquets of Pittsburgh bituminous slack. The strong characteristic odor of this material may be an objection to its use.

8. Two per cent of cell pitch made strong briquets with Pittsburgh bituminous slack. As this material is soluble in water, it would not make briquets suitable for storage in the open but if stored under cover the briquets will stand up indefinitely. The effect of moisture on this binder being detrimental to its binding qualities, it is not surprising that 8 per cent was required to briquet lignite from Texas and North Dakota, whereas 4 per cent was sufficient to briquet a sample of Utah subbituminous coal. Six per cent was required to briquet Philippine lignite and Washington subbituminous coal. The briquet having the greatest compressive strength was made of Pittsburgh slack and 6 per cent of cell pitch. The briquet broke at a pressure of 3,800 pounds per square inch, whereas a 6 per cent water gas pitch briquet of this coal broke at a pressure of 2,220 pounds per square inch.

9. Three per cent of sulphite liquor was sufficient to briquet Pittsburgh slack and 5 per cent was sufficient to briquet anthracite culm. Less satisfactory results were obtained by mixing this material with lignites. It was found that 9 per cent of this binder was not sufficient to briquet Texas, North Dakota, or Philippine lignite. The moisture in these fuels seemingly affects the binding qualities of this material.

SULPHITE PITCH AS A BINDER.

DERIVATION OF THE PITCH.

A new binding material called "sulphite liquor" has been suggested for briquetting, but more recently a product called "sulphite pitch," obtained from this sulphite liquor, has attracted considerable attention and obviously possesses valuable properties as a binder.

In the sulphite process of manufacturing paper pulp the wood under pressure is boiled with sulphurous acid, or with acid sulphite of calcium and magnesium. The action of sulphurous acid under pressure and at a high temperature upon the lignin and other incrusting matters of the wood fiber is probably a hydrolysis; that is, the complex molecules of the lignin, etc., are broken down, and the resulting products, largely organic acids and aldehydes, become soluble in the liquor.

The waste liquors are light brown and contain much matter extracted from the wood. Until recently they had no commercial value and their proper disposal was often a serious matter, as they polluted streams into which they were emptied, killing the fish.

PREPARATION AND USE IN GERMANY.

Several patents have been obtained for converting the waste liquors into fertilizers, adhesive material, or other commercial products; but the most promising way of utilizing them at present seems to be by making a concentrated product known as "sulphite liquor," or a solid form known as "sulphite pitch" or "cell pitch" (selpech).

The method of preparing sulphite liquor or cell pitch from these wastes, according to W. Sembritzki,^a manager of the Walsum paper mills (Germany), is as follows:

At the Walsum factory the liquors have a specific gravity of 1.05 and an acidity, calculated as sulphur dioxide, of 0.32 per cent. The thick sirup, density 35° B., from the evaporators has the following composition: Water, 28.88 per cent; insoluble matter, 13.96 per cent; extract, 57.16 per cent; tanning matters, 22.96 per cent; nontannins, 34.24 per cent. The solid cell pitch contains: Water, 13.76 per cent; soluble extract, 86.60 per cent; tanning matters, 31.84 per cent; nontannins, 54.76 per cent. The nontannins of this sirup consist of organic matter, 24.76 per cent; mineral matter, 9.44 per cent. The nontannins of the cell pitch consist of organic matter, 36.08 per cent; mineral matter, 18.68 per cent.

If the evaporators are made of iron, it is necessary to neutralize the liquor with lime before concentration, and the incrustation of the tubes with calcium compounds is somewhat troublesome. If copper evaporators are used, the liquors may be concentrated in their acid state. If the concentrated liquor is to be used for briquetting flue dust, neutralization with lime is essential to the proper working of the briquets in the blast furnace. If, on the other hand, the concentrated liquors are intended for tanning purposes, neutralization is a drawback, as the calcium compounds make the leather brittle.

The success of the Walsum experiments is attributed in large part to the sextuple-effect Kestner evaporator used. Each effect has a heating surface of 55 square meters; the first vessel is heated with steam at 3 to 4 atmospheres and the boiling point in the last effect is 50° C. The liquor requires about three hours to reach a concentration of 35° B. The sirup is converted into solid pitch by two steam-heated drums which dip below its surface. The first drum concentrates the sirup from 35° B. to 60° B.; the second converts it into a solid film, which is removed with a scraper. Ten kilograms of waste liquor yield 1 kilogram of dry pitch. The fuel consumption is 1 ton of coal per ton of pitch. It is stated that a selling price of \$9.50 a ton would show a good profit.

The pitch has the appearance of black opaque resin, but is quite soluble in water. It is used as a binder for fuel and ore briquets. By its use the dust from blast furnaces, containing 40 per cent of iron and hitherto a waste product, has been formed into briquets for remelting. It has also been used as a dressing for coarse canvas, etc. Large quantities of the sulphite liquor, having a density of about 35° B., are shipped from Germany to England in iron drums. The liquor is used in England as a binder for the sand cores and molds used in casting, and is sold there at \$12.50 a ton.

^a Sembritzki, W., Preparation of cellulose pitch from waste sulphite liquor: *Wochbl. Papier-fabr.*, 1908 (39), p. 658.

In a report made in 1909 George E. Eager,^a United States consul at Barmen, Germany, states that sulphite pitch is intensely glutinous and its binding power high. According to Mr. Eager, 7 to 10 per cent of coal tar is needed in briquetting bituminous coal, but the same results can be obtained by the use of 5 per cent of sulphite pitch, and some kinds of coal or ore can be briquetted by using only 2 to 3 per cent. Sulphite pitch burns without smoke or odor and is an ideal fuel for the household, as well as for industrial purposes. The use of briquets made with this sulphite pitch will help to solve the smoke problem in cities. Coke briquets made with this new binder have been tried in blast furnaces and on torpedo boats with promising results. Sulphite pitch does not soften under heat and burns at a high temperature. It can be ground to any consistency; it can be obtained at a low price where there are cellulose mills. Anthracite briquets for household use manufactured with sulphite pitch burn without smoke or odor.

Regarding other uses of sulphite pitch and its composition, the report states:

Recent attempts to briquet coke gravel and dross show that the resultant briquet can be considered a perfect substitute for coke. Practical trials of the briquets in both blast and cupola furnaces have shown that the briquets do not fall to pieces under the highest temperature, but burn while gradually shrinking. Fine ore, bog-iron ore, brown ore, manganese ore, oxide, furnace cadmia (iron dust from blast furnaces), and other ores can all be briquetted by the use of sulphite pitch and successfully melted in the furnace. All attempts to briquet the above materials with coal-tar pitch have failed, because the binding agent burned at a lower temperature, leaving the material in dust, as before. With sulphite pitch it is possible to briquet furnace cadmia so that it can be melted in a blast furnace, making possible a great saving to the iron industry.

In general, sulphite pitch consists of the following substances: Fixed carbon, 25 to 35 per cent; volatile matter, 50 to 60 per cent; ash, 8 to 12 per cent; and water, 10 to 15 per cent. The percentage of ash can be materially reduced. Through the origin of sulphite pitch its ashes contain sulphur up to 20 per cent, or 2.5 per cent of the sulphite pitch. The sulphur is combined with iron and lime, so that the sulphur remains in the ashes and can not do any damage. Sulphite pitch is soluble in water, and briquets made from it are not waterproof. The sulphite-pitch briquet is, however, more waterproof than the lignite briquet, the making of which has become a flourishing industry. The sulphite briquet is not hygroscopic and can be made absolutely waterproof by a special treatment.

The production of sulphite pitch, as well as its use in the process of briquetting, call for special processes and machinery which have taken years of expensive experiment to develop successfully, and

^a Daily Consular and Trade Reports, No. 3361.

both the material and its use for briquetting are patented in all the principal countries.

Sulphite pitch will not in any way interfere with coal-tar pitch in its use for briquetting bituminous coals, but the superiority of sulphite as a binding agent makes possible its use in the briquetting of harder coals and cokes, also iron dust and ores.

THE POLLACSEK BRIQUETTING PROCESS.^a

The Pollacsek briquetting process, which employs the waste liquor of the sulphite paper-making process as a binder, has been adopted by the Hungarian Government for its collieries, and that Government has erected the machinery for the first plant. It is estimated that a plant in the United States producing 10 tons daily would cost about \$5,000 and that the cost of operation would not exceed 60 cents a ton and might be as low as 40 cents in favored places.

Fine coal or slack is used, pieces above a quarter of an inch being crushed. Such crushing is necessary, because any larger pieces of coal would be shattered in the molds of the press, and as there would be no binder between the shattered fragments, the briquets would not be strong enough to stand transportation.

The inventor has a process by which the waste liquor of the pulp mills may be evaporated to a sirup or reduced to a dry powder for convenience in transportation. In the United States the removal of free acid is accomplished by the use of lime. The evaporated sulphite liquor has a sirupy consistency and considering its price and efficacy is the best core binder for foundry use known to-day. It is probable that about the same method of removing free acid is used in the Pollacsek process.

In view of the fact that in making fuel briquets only 3 to 5 per cent of this binder is used, its cost per ton of coal briquetted is small. Another point in favor of the process is that by utilizing the waste liquor from the pulp mills it protects streams from pollution by the liquor.

If the binder is in the form of a sirup, it is diluted; if in the form of solid pitch, it is dissolved in the proper quantity of water. From the mixer the briquet material passes into a rotary kiln, where it is heated and partly dried. The success of the process depends on maintaining the right temperature, for the product goes directly from the kiln to the briquetting press. If the temperature varies only 10° F. either way from the standard, the briquets are not hard or strong enough to stand transportation from the press.

If the kiln dries the material enough and the temperature in the kiln is kept at the right point, the coal will stick together when pressed, but will not adhere to the molds.

^a Information furnished by Richard Moldenke, consulting engineer.

If the briquets are not to be waterproofed, they go through a drying oven directly to the cars for shipment. If they are to be waterproofed, they pass from the drying oven through an emulsion of bitumen and water and again through a drying oven, whence they are delivered to the cars. As little labor is needed and the cost of binder is small, claims of great cheapness are made for the process. The secrets of the process are the proper mixing of the coal and binder, the proper control of the kiln temperature, and the weatherproofing of the briquets.

The briquets made by the Pollacsek process are hard, stand rough handling, and are sufficiently weatherproof for every purpose.

The briquetting section of the Bureau of Mines has made laboratory tests with this sulphite pitch, the results of which are published in this bulletin on previous pages.

APPARATUS FOR BRIQUETTING COAL AND COKE WITH CELL PITCH.

The briquetting of coal and coke (with cell pitch binder) was first carried on by "Hengstenberg & Benzenberg, Brikettwerk m. b. H." at Ruhrort, and later at Ruhrrevier and Stettin, Germany.

For the production of cell-pitch briquets essentially the same apparatus is used as for briquetting with coal-tar pitch. Of especial importance is the best possible mixing and kneading of the binding material to be briquetted with superheated steam. Large-sized briquets are pressed with a high pressure, and for this purpose presses are built sufficiently strong to exert a pressure of 7,000 to 8,500 pounds per square inch. For the smaller briquets a lower pressure can be used. Egg-shaped briquets can be produced with cell pitch on the same presses on which coal-tar pitch briquets are made.

WEATHERPROOFING TREATMENT.

Weatherproofing is accomplished by heating the briquet or by a previous treatment of the briquetting material.

The after-heating is the older method and consists of partly coking the sulphite pitch. This produces a briquet that is waterproof and weatherproof, as well as unusually hard and firm and rough on the surface. The method is in many cases economical and worth practicing if a cheap source of heat is available. The briquets, dried by blast-furnace gases at a temperature of 300° C., furnish weather-resisting coke briquets.

The latest method of waterproofing sulphite-pitch briquets consists of the addition of certain chemicals (at the present time a secret) directly to the briquet material before pressing. The additional cost is about 35.7 cents per ton of finished briquets. This increased cost should be less if the chemicals can be bought more cheaply in consequence of an enlarged use for briquetting.

APPLICATION AND COST OF PRODUCTION.

The cost in Germany of building a two-press plant having an hourly capacity of 5 tons and producing yearly 60,000 tons is as follows:

Cost of two-press cell-pitch briquetting plant in Germany.

Item.	Marks.	Dollars.
Ground.....	30,000	7,140
Building, including foundation and building stone.....	50,000	11,900
Machinery, power, lighting, etc.....	175,000	41,650
Tools, etc.....	10,000	2,380
Spar track.....	25,000	5,950
Miscellaneous.....	5,000	1,190
Total.....	295,000	70,210

The cost per ton of briquets, including cell pitch but not the raw material, on the basis of a production of 60,000 tons per year, is as follows:

Cost per ton of briquets, including sulphite pitch.

Percent- age of sulphite pitch added.	Cost per ton.	Percent- age of sulphite pitch added.	Cost per ton.
4	\$0.72	6½	\$0.96
4½	.77	7	1.008
5	.82	7½	1.056
5½	.864	8	1.104
6	.912		

The cost of the after treatment of the briquets to make them water-proof must be determined separately for each case. It is estimated, however, at present as not more than 36 cents per ton.

To this cost of briquetting and cost of after treatment are also to be added the cost of the raw material, as well as the royalty. Using as a basis plants already operating in Germany, a royalty of 1 mark or 24 cents per ton of briquets is assumed.

**CELL PITCH FOR BRIQUETTING FINE ORES, FLUE DUST, AND
THE LIKE.**

Cell pitch possesses manifest advantages over coal-tar pitch for briquetting fine ore, flue dust, etc. Its solubility in water does not cause disintegration of the briquets by the steam present in the flue of the blast furnace. The sulphite-pitch ore or flue-dust briquets do not fall to pieces, and the small sulphur content of the binding material exerts no bad effect.

CELL-PITCH EXPERIMENTS OF DR. TRAINER.

The results of cell-pitch experiments by Dr. Trainer (Gewerkschaft Eduard) are essentially the same as those in the above-described bituminous and coke briquetting by means of cell pitch. The briquetted material, after the addition of the required quantity of sulphite pitch, was treated thoroughly in a mixing or kneading machine with superheated steam and was immediately pressed under the high pressure of 7,000 to 8,500 pounds per square inch. The cost of installing a two-press plant for fine ore or flue dust is about as high as for fuel, namely, about \$70,210. The briquetting cost for ore briquets, in consequence of their high specific gravity which allows a larger capacity of the press by weight, is about \$0.13 less per ton of briquets than for coal briquets, as shown in the following table:

Cost per ton of briquets, with various percentages of sulphite pitch.

Percent- age of sulphite pitch.	Cost per ton.	Percent- age of sulphite pitch.	Cost per ton.
4	\$0.59	6½	\$0.53
4½	.64	7	.88
5	.69	7½	.93
5½	.73	8	.97
6	.78		

It is only in rare cases that it is necessary to waterproof and weatherproof ore briquets. When this is necessary the cost of the briquetting is increased 24 to 36 cents per ton.

The cell-pitch method has found important application in the iron works of the Gewerkschaft Deutscher Kaiser in Bruckhausen, near Ruhrort. An experimental briquetting plant was erected in 1908 to furnish cell pitch at cost to the "Vereinigten Gewerkschaften Eduard und Pionier," Walsum on the Rhine. The results were so favorable that the Gewerkschaft Deutscher Kaiser entered into an agreement with the above firm to furnish sulphite pitch for the briquetting of 400 tons of flue dust per day. The flue-dust cell-pitch briquets are produced with an addition of 4½ per cent of sulphite pitch. The disintegration of the briquets and the small sulphur content of the binding material have not yet proved to have injurious effects. The briquets are completely reduced, and the ore in the same blast-furnace zone is favorably influenced. If, for example, 15 per cent of briquets is added to the charge, and an equal percentage of ore omitted, there is a decrease in flue dust of 30 per cent, as well as an increased production of iron with a smaller use of coke.

LITERATURE ON SULPHITE LIQUOR AND SULPHITE PITCH.

A brief list of literature relative to sulphite liquor and sulphite pitch is presented below:

- Ahrens, F. B., Utilization of waste sulphite-cellulose lyes: *Chem. Zeitschr.*, 1905, pp. 40-41; abstracted in *Jour. Soc. Chem. Ind.*, 1905, p. 343.
- Krause, H., Contribution to the chemistry of sulphite wood pulp waste liquors: *Chem. Ind.*, 1906 (29), pp. 217-227; abstracted in *Jour. Soc. Chem. Ind.*, 1906, p. 493.
- Robeson, J. S., U. S. Patents 833635, 851378, 851379, 851380, 851381; abstracted in *Jour. Soc. Chem. Ind.*, for 1906 and 1907.
- Sembritzki, W., Preparation of cellulose pitch from waste sulphite liquor: *Wochbl. Papier-fabr.*, 1908 (39), p. 658; abstracted in *Chem. Abstracts, Amer. Chem. Soc.* 1908, p. 1885; Manufacture of pulp pitch from waste sulphite liquors and its use in briquetting: *Wochbl. Papier-fabr.*, 1908 (39), pp. 2866-2867; abstracted in *Jour. Soc. Chem. Ind.*, 1908, p. 915.
- Knoesel, T., Manufacture of pulp pitch from waste sulphite liquors and its use in briquetting: *Wochbl., Papier-fabr.*, 1908 (39), pp. 3276-3278, 4049.
- Eager, George E., *Daily Consular and Trade Reports*, No. 3361; also published in *Power and the Engineer*, March 9, 1909, p. 447.
- Thorp, F. H., *Outlines of industrial chemistry*, 1899, p. 503.
- Franke, G., *Handbuch der Brikettbereitung*, vol. 2, 1909, pp. 72-80.

TESTS TO WHICH BRIQUETS WERE SUBJECTED.**PHYSICAL TESTS.**

Samples of the better lots of briquets made were subjected to the drop test, the compression test, and the tumbler test to determine their cohesive strength and to show their ability to endure handling; absorption, specific-gravity, and weathering tests were also made.

DROP TEST.

In the drop test 50 pounds of briquets was placed in a drop-bottom box 24 inches square and 12 inches deep inside, supported 6 feet above a concrete floor. The briquets were dropped on the floor; the pieces were gathered up and those that were held by a 1-inch screen (square holes) were returned to the box and dropped again. The floor was swept clean before each drop. This procedure was repeated five times, after which the weight of pieces that would not pass through the screen was determined. The results of the test were found to agree closely with those of the tumbler test, although no attempt was made to standardize the two tests, one being a shatter test, the other being more on the order of an abrasion test.

TUMBLER TEST.

In the tumbler test a weighed quantity of the briquets (about 50 pounds) was placed in a horizontal sheet-steel cylinder, 2 feet in diameter and 3 feet long, and rotated 2 minutes at a uniform speed of 28 revolutions per minute. The contents of the tumbler were then

sized by a 1-inch mesh screen, and the part that passed through was screened through a 10-mesh sieve. The weight of the pieces held by each screen was determined and the "percentage held" computed and reported as in the drop test.

ABSORPTION TEST.

The object of the absorption test was to determine (1) the rate of absorption of water by the briquets each day; (2) the time required for the absorption to become practically complete; and (3) the time at which the absorption actually ceased or the briquet disintegrated. The apparatus consisted of a hydrostatic balance with containing tank and four galvanized-iron pans 24 by 36 by 6 inches.

Four Renfrow briquets, two German briquets, or one English briquet were taken as a sample for each representative lot of briquets tested. These briquets had been stored under cover directly after being made, so they may be safely considered as air-dried samples. Whenever possible, briquets with perfect surfaces were used.

The method used in testing was as follows:

(a) The sample was weighed in air on the upper part of the hydrostatic balance. (b) The sample was then weighed in water on the lower shelf of the balance, the hour and minute of this immersion being noted. (c) The sample was removed from the balance, placed in pans, and covered with one-half inch of water. (d) Each succeeding day at the same time as the first immersion the operation noted in b was repeated. (e) The gain in weight was noted each day by subtracting the observation for the previous day from the weight noted in operation b. (f) This gain in weight was calculated to a percentage of the original weight of the dry briquet, and so recorded each day.

DETERMINATION OF DENSITY.

The apparent specific gravity, being the weight in air divided by the loss of weight in water, was calculated from readings *a* and *b* of the absorption tests. In the earlier tests fragments from the compression tests were used, each piece weighing about 65 grams. These earlier tests were made with a Nicholson hydrometer.^a

WEATHERING TEST.

To determine the comparative weather-resisting qualities of the briquets and the samples of fuel from which they were made, small piles of each lot of satisfactory briquets and lumps of the raw fuel were exposed to the action of sun, wind, and rain. Plates XVI to

^a See Lord, N. W., Experimental work conducted in the chemical laboratory of the United States fuel-testing plant at St. Louis, Mo., Jan. 1, 1905, to July 31, 1906: Bull. 23, Bureau of Mines, pp. 23-28 (reprint of U. S. Geol. Survey Bull. 323).

XXI, made from photographs taken of Pittsburgh tests at various intervals up to 286 days, show the different stages of weathering. The length of time the briquets were exposed before examination, and the condition of the briquets when examined, are stated in the tabulated results of tests. The key to the conditions designated A, B, C, D, and E is that originally stated in Bulletin 332^a of the United States Geological Survey, and is as follows:

Condition A: Briquets practically in same condition as when put out. Surfaces show no signs of erosion or pitting. Briquets hard, with sharp edges, and fracture same as that of new briquets.

Condition B: Shape of briquets unchanged. Surfaces of those on top of pile have lost luster, with evidences of pitting; corners and edges worn off by erosion. All briquets firm, with fracture practically the same as that of new briquets.

Condition C: Top briquets appear similar to those in condition B, and show signs of further disintegration, having lost original sharp fracture. Erosion more evident on all briquets on outside of the pile. Inside briquets still firm, retaining original characteristics.

Condition D: Top briquets so badly disintegrated that they crumble to pieces on handling. Briquets in center of pile show signs of disintegration; luster of surfaces gone, edges soft, and break easily in the hand. Fracture not so sharp as when newly made, but briquets firm and handled without breaking.

Condition E: Entire pile disintegrated. In many cases the only briquets retaining their original shape are those protected from the weather. Briquets can not be handled safely, but crush easily in hand.

COMPRESSION TEST.

In order to obtain some standard of comparison between the briquets, the crushing strength of many of them was determined on a 200,000-pound Olsen machine. When only a 1-ton lot was made the best briquets were selected for these tests, as it was reasonable to suppose that in making larger quantities the average briquet would be equal to the better ones made in small lots. Of the 5 to 15 ton lots, the average briquets were taken.

The briquets were tested on the machine lengthwise, so that their position gave a uniform area of 28.7 square inches, the cross-section area of the molds. The thickness of the briquets varied from 5.75 to 6 inches, depending on the pressure. Any differences due to lack of parallelism between the ends were automatically corrected by the swiveling head of the machine. Pieces of millboard were placed between the briquets and the steel. The slowest speed of compression was the most satisfactory.

^a Report of the United States fuel-testing plant at St. Louis, Mo., Jan. 1, 1906, to June 30, 1907, J. A. Holmes in charge. 1908. 299 pp.

A comparison of the compression tests of briquets made of anthracite coal and briquets made of anthracite and soft coking coal shows that the crushing strength of the latter is less, although it stands the weather somewhat better than the anthracite briquet with pitch alone.

The coke-breeze briquets were far stronger than any others made, as is indicated by the results of the compression tests.

Although the results of the compression tests are to some extent comparable one with another, they do not give the desired information regarding the actual strength or endurance of the briquets, for in some instances the crushing strength is rather high, whereas the briquets themselves could not have been handled to any great extent without breaking. The most satisfactory test for determining the strength of the briquets is afforded by the drop and tumbler tests. These give results that may be compared, and determine the relative strength of the briquets and the amount of handling they will stand in transportation.

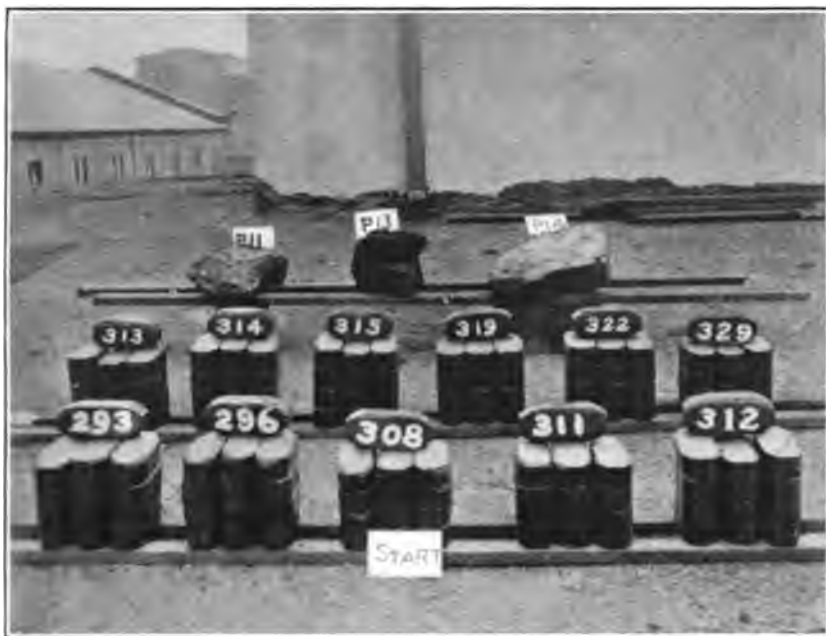
The compression test was discontinued at the St. Louis plant, as the results from briquets made of the same coal and under similar conditions did not check. This difference was probably due to a lack of uniformity in the composition of the various briquets of a test. In the laboratory tests made at Pittsburgh, however, the compression test was found useful to determine the comparative strength of briquets made on the hand experimental press, as in these tests conditions of heating and mixing could be closely controlled, and only five briquets weighing less than one-half pound each were made for each test. This number was too small for the drop or tumbler tests. These briquets, being flat-ended cylinders 3 inches in diameter and about $1\frac{1}{4}$ inches thick, were suitable for making compression tests on the hydraulic press on which they were formed by removing the mold and substituting flat plates. The results obtained from these tests of laboratory briquets agreed with sufficient closeness to determine the relative strength produced by different percentages of binder.

CHEMICAL TESTS.

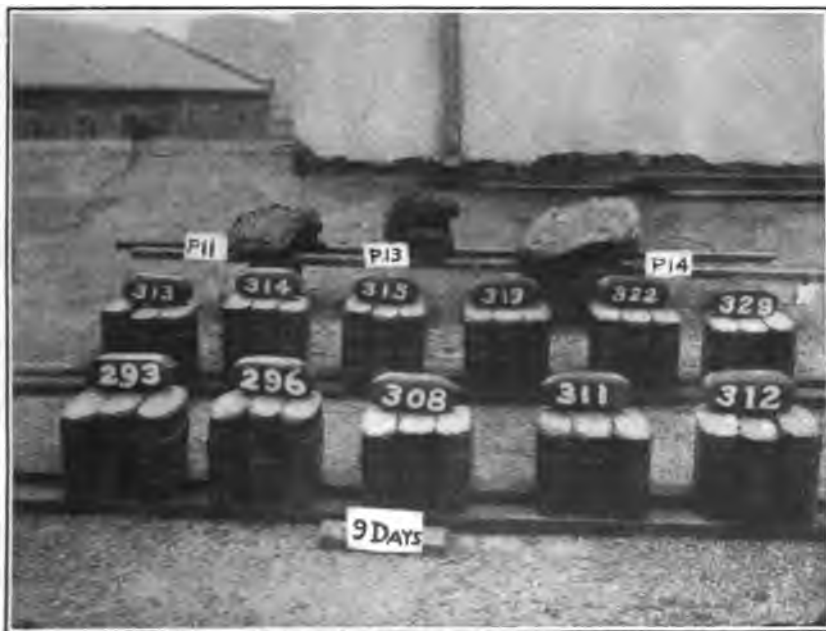
Chemical analyses were made of the fuels and binders used and the briquets manufactured. In most cases proximate analyses only were made of the briquets. Ultimate analyses, as well as proximate analyses, were generally made of the briquetted fuels.

PROXIMATE AND ULTIMATE ANALYSES.

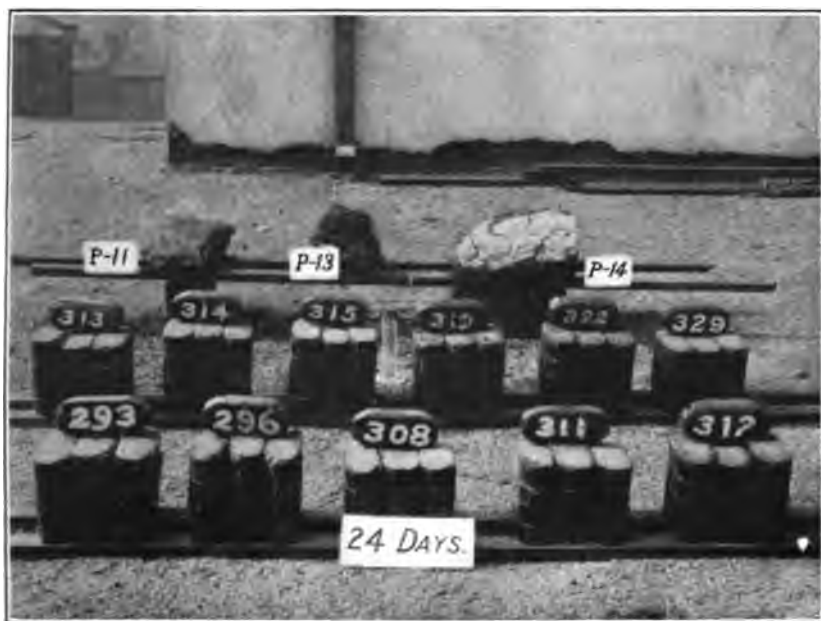
The proximate analyses (including sulphur) herein reported are in terms of "as received." That is, the results of the analyses were calculated on the sample with the moisture in it as received at the



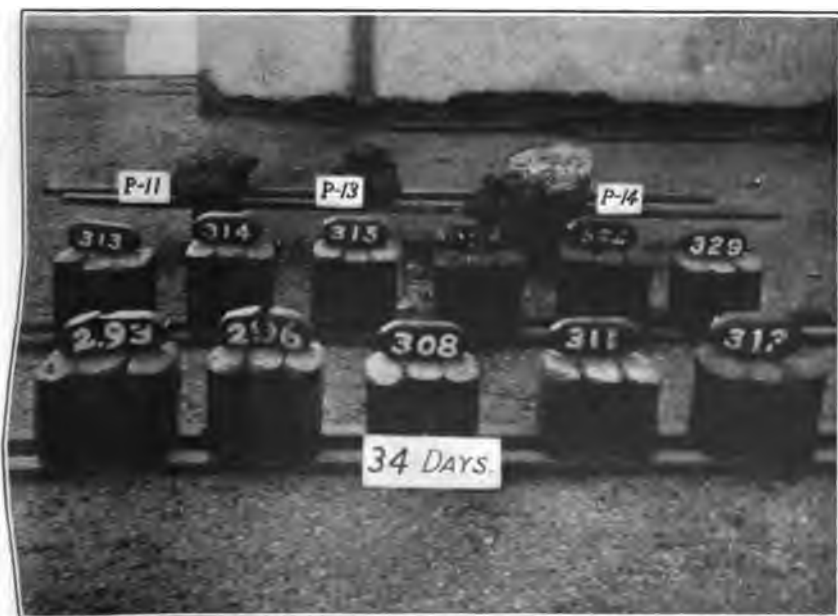
A. BRIQUETS AT BEGINNING OF WEATHERING TEST.



B. BRIQUETS AFTER EXPOSURE FOR 9 DAYS.



A. BRIQUETS AFTER EXPOSURE FOR 24 DAYS.



B. BRIQUETS AFTER EXPOSURE FOR 34 DAYS.



A. BRIQUETS AFTER EXPOSURE FOR 66 DAYS.



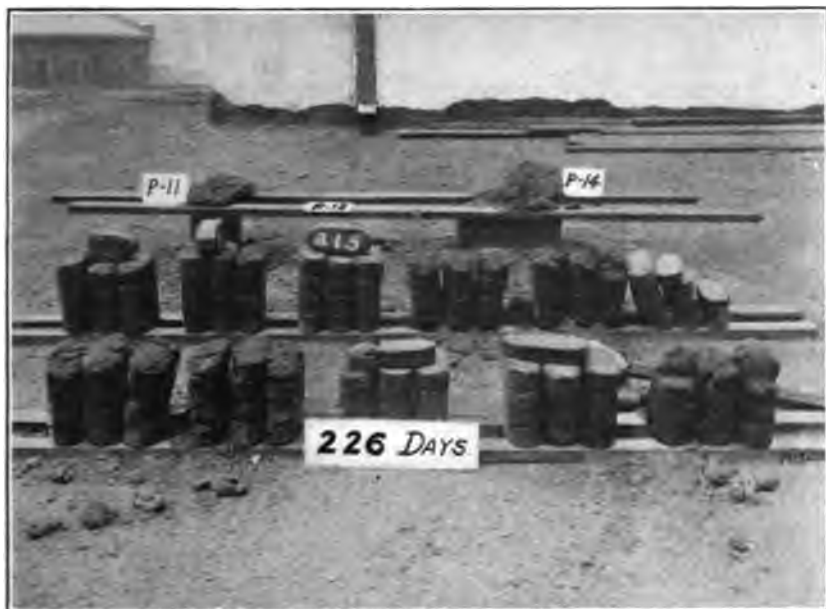
B. BRIQUETS AFTER EXPOSURE FOR 96 DAYS.



A. BRIQUETS AFTER EXPOSURE FOR 126 DAYS.



B. BRIQUETS AFTER EXPOSURE FOR 166 DAYS.



A. BRIQUETS AFTER EXPOSURE FOR 226 DAYS.



B. BRIQUETS AFTER EXPOSURE FOR 286 DAYS.

laboratory and not on the basis of the dry coal. The ultimate analyses herein reported are on a moisture-free basis, as in this form the analyses are more useful for comparison and future study.

MOISTURE TEST.

In carrying out the lignite tests at Pittsburgh it was necessary to make rapid and frequent moisture determinations for controlling the drying of the fuel. As the usual methods of determining moisture were not rapid enough a special method was devised by F. M. Stanton, chemist in charge of the fuel laboratory. The procedure was as follows: Fifty grams of coarsely crushed lignite was placed in a distilling 200-c. c. flask, covered with 70 c. c. of kerosene, and heated over a gas flame. The moisture driven off was condensed in a Liebig condenser and run into a tall 25-c. c. measuring cylinder or burette. The distillation was carried on until a thermometer in the flask showed a final temperature of 350° F. The quantity of water in the measuring cylinder was noted when the thermometer in the flask registered 250° F., 300° F., and 350° F., and the percentage of moisture in the sample was taken as the percentage of moisture distilled at 300° F. Of course, some of the kerosene distilled over, but it readily separated from the water, and the quantity of the latter in the measuring cylinder could be accurately read.

EXTRACTION TEST.

To determine the percentage of bitumen or natural binding material in the raw or briquetted fuel, samples were extracted with carbon bisulphide in a Soxhlet apparatus in the following manner:

Five grams of the finely ground material was weighed in an extraction thimble and extracted in a flask that had been dried for one-half hour at 108° C., and was cooled in a desiccator and weighed. After the material had been in the extractor five hours the carbon bisulphide in the flask was distilled and recovered in the Soxhlet extractor; the flask and its contents were dried at 108° C. for one hour, cooled, and weighed, and the percentage of material extracted was calculated from the weight of the residue.

From the results of the extraction tests of fuel, bituminous binder, and briquets it was possible to calculate the percentage of pitch binder present in the briquet by the use of the following equation:

$$x = \frac{100 (A - B)}{C - B}$$

in which x is the percentage of pitch binder present; A is the extraction figure of the briquet, calculated to an "as-received" basis; B is the extraction figure of the coal used, calculated to an "as-received" basis; C is the extraction figure of the pitch binder used, calculated to an "as-received" basis.

COMBUSTION TESTS.

Combustion tests were made of the fuels, binders, and briquets. Heat-value determinations were made by the chemical laboratory using the Mahler bomb calorimeter. The heating value of the materials was also calculated from their analyses, but as the Mahler bomb furnishes more reliable data on the heat value, the results of the bomb tests only are included in this report. The heat values in this report are on a moisture-free basis unless otherwise noted.

STEAMING TESTS.**STATIONARY-BOILER TESTS.**

The only method available at St. Louis for determining the relative heating values of briquets consisted in burning them under boilers, the conditions being uniform with those for the steaming tests of coals. These tests were made by the boiler section in every case where there was a sufficient number of briquets, and are reported under "steaming tests" in this bulletin. When coal was not available to make on each machine enough briquets, with the same percentage of binder, for a complete steaming test on both the English and Renfrow briquets, the test was divided into five-hour runs for each machine.

Several small lots of briquets containing various percentages of binder were burned under the boiler to show the behavior of the briquets in the fire and the effect on the quantity of smoke produced, but the main test was of briquets of a uniform percentage of binder. In many tests it was necessary to break up the English briquets in order to obtain capacity on the boiler, although every effort was made to burn them unbroken for comparative results.

Besides these tests of briquets at the plant, two series of tests were run—one using the briquets in locomotive boilers and the other in boilers for house heating. In many tests the product was divided, part being used for a house-heating test and part for a locomotive test.

The combustion tests made with briquetted fuel in stationary, locomotive, and house-heating boilers, and the briquetting test number from which the briquets were obtained, are shown in the table following. This table furnishes a key to the composition, methods of making, and qualities of the briquets used. The steaming tests mentioned in the third column under "Regular steaming test No." are reported in Bureau of Mines Bulletin 23.^a The locomotive tests are listed in the fourth column. The Seaboard Air Line locomotive

^a Breckenridge, L. P., Kreisinger, Henry, and Ray, W. T., *Steaming tests of coals and related investigations*, Sept. 1, 1904, to Dec. 31, 1908. 1912. 380 pp., 2 pls., 94 figs.

tests mentioned are described in Bureau of Mines Bulletin 23,^a mentioned above, and in Bulletin 34, Bureau of Mines.^b The locomotive tests of the Pennsylvania 18 (Lloydell) coal are described in Bureau of Mines Bulletin 37.^c The locomotive tests made by the Atlantic Coast Line Railroad Co. and the Chesapeake & Ohio Railway Co. are also described in Bulletin 37, mentioned above.

The steaming tests of briquets in house-heating boilers are described in Bureau of Mines Bulletin 27.^d The steaming tests of briquets on the U. S. torpedo boat *Biddle* are described in Bulletin 33 of the bureau.^e

KEY TO STEAMING TESTS.

A résumé of the locomotive, house-boiler, and torpedo-boat tests is included in this bulletin for the benefit of readers who may not have copies of the bulletins mentioned above.

Key to steaming tests of briquets.

Field designation.	Briquet- ting test No.	Briquets used in—		
		Regular steaming test No.	Locomotive test No.	House boiler test No.
Alabama 2.....	131	410		
Alabama 4.....	123	413		
Argentina 1.....	180	485		
Arkansas 7.....	106			
Arkansas 13.....	164			45
Do.....	214			
Do.....	221			
Illinois 7E.....	244			48
Illinois 9C.....	189	492		
Do.....	190	497		
Do.....	233			
Do.....	234			39
Illinois 12BW.....	166	463	25, 26, 27	
Do.....	177		28, 29, 32	13
Do.....	181		30, 31	
Illinois 20, mixed with Kentucky 2B.....	103			
Do.....	104			
Illinois 21.....	113	318		
Illinois 23.....	116	321		
Do.....		322		
Illinois 28A.....	140			
Do.....	141		1, 4	
Do.....	142		2	
Do.....	143		3, 13, 15, 19	
Do.....	162	459		
Do.....	163	459		
Illinois 28B.....	144		16, 17, 18, 20	
Do.....	158			
Do.....	159		21, 22	
Do.....	160			
Do.....	161		14	
Illinois 29A.....	170	465		
Do.....	171	465		9
Illinois 29B.....	175	466		12
Illinois 30.....	225	511		
Do.....	226	511		
Do.....	228			

^a Pp. 321-329.^b Ray, W. T., and Kreisinger, Henry, Tests of run-of-mine and briquetted coal in a locomotive boiler. 33 pp., 9 figs. (reprint of U. S. Geol. Survey Bull. 412).^c Coon, W. F. M., Comparative tests of run-of-mine and briquetted coal on locomotives. 58 pp., 4 pls., 35 figs. (reprint of U. S. Geol. Survey Bull. 363).^d Randall, D. T., Tests of coal and briquets as fuel for house-heating boilers. 44 pp., 3 pls., 2 figs. (reprint of U. S. Geol. Survey Bull. 366).^e Ray, W. T., and Kreisinger, Henry, Comparative tests of run-of-mine and briquetted coal on the torpedo boat *Biddle*. 50 pp., 10 figs. (reprint of U. S. Geol. Survey Bull. 33).^f Briquets from this test were used in water-gas machine test.

Key to steaming tests of briquets—Continued.

Field designation.	Briquet- ting test No.	Briquets used in—		
		Regular steaming test No.	Locomotive test No.	House boiler test No.
Illinois 31.....	185	491		
Do.....	186	489		
Do.....	224			33, 34
Do.....	237			44
Illinois 33.....	206			
Do.....	207			
Do.....	210			
Do.....	235			43
Indiana 1B.....	217		48, 49, 51	37
Indiana 5B.....	229		47	
Do.....	230		46	36
Indiana 6B.....	220		45, 50	
Do.....	227			38
Indiana 19.....	165	484	23, 24	
Indiana 20.....	169		52	
Indian Territory 2B.....	135			
Do.....	136			
Do.....	137			
Do.....	138	456		
Do.....	139			
Do.....	145			
Do.....	146			
Do.....	147			
Do.....	148	453	5, 6, 10, 11, 12	11
Do.....	149		9	
Do.....	157	456		10
Do.....	168			
Indian Territory 2C.....	153	455		
Do.....	154	455		
Do.....	155	455		
Do.....	156	455		
Indian Territory 9.....	167	450		
Kansas 2B.....	182	487	39	40
Do.....	183	488	38	1
Do.....	194	495	36	
Do.....	195			
Do.....	199		33, 34	
Do.....	203		37	
Do.....	204		35	
Kentucky 2B.....	102		{ Big Four R. R. prelim- inary tests. }	
Maryland 2.....	191	493		
Do.....	192	493		
Do.....	193	493		
Do.....	231	518		21, 22, 23
Missouri 10.....	178	486		2, 3, 4, 5
Do.....	179	486		
Do.....	241		59, 61	
Do.....	245		{ 57, 58, 62, 63, 67, 68 }	
Do.....	246		63	
Pennsylvania 15.....	176	467		
Pennsylvania 15, mixed with Rhode Island 1.....	184			47
Pennsylvania 16.....	172	468		
Do.....	173	468		
Do.....	174	468		
Pennsylvania 18.....	196		79, 90	
Do.....	197	515	{ 72, 73, 74, 75, 76, 87, 89 }	
Do.....	200			
Do.....	201	499		
Do.....	202	499		
Do.....	205			
Do.....	232		{ 78, 80, 81, 82, 86, 88 }	28, 29
Do.....	236			30, 31
Do.....	250			46
Pennsylvania 18 and Rhode Island 1.....	243			35
Pennsylvania 18 and Miscellaneous 9.....	238			49
Do.....	239			50, 51
Do.....	240			56
Do.....	248			
Pennsylvania 19.....	218	508		
Do.....	219	508		
Do.....	242			

Key to steaming tests of briquets—Continued.

Field designation.	Briquet- ting test No.	Briquets used in—		
		Regular steaming test No.	Locomotive test No.	House boiler test No.
Pennsylvania 20.....	198			26, 27
Do.....	208	514		
Do.....	209	514		
Do.....	212			
Do.....	213			41, 42
Do.....	215	512		
Do.....	216	512		
Pennsylvania 22.....	211			16, 17
Do.....	222	510		
Do.....	223	510		18, 19
Rhode Island 1 and Utah 1.....	127			
Rhode Island 1 and Utah 2.....	133			
Rhode Island 1 and Pennsylvania 15.....	184			47
Rhode Island 1 and Pennsylvania 18.....	243			35
Tennessee 1.....	130	409, 411		
Tennessee 4.....	120	405		
Tennessee 4 and Miscellaneous 5.....	150			
Do.....	151			
Tennessee 7A.....	121	406		
P-11 ^a	322			H86, H87
P-14 ^a				H86, H89

Field designation.	Briquet- ting test No.	Briquets used on—		
		Torpedo boat, <i>Biddle</i> test No.	Seaboard-loco- motive test No.	House boiler test No.
J-4.....	260	4, 5, 6.....		(b)
J-9.....	272	17, 18, 19.....		
J-11.....	276	10, 11, 12, 13, 20.....		
J-12.....	286		5, 6, 7, 8.....	
J-13.....	287		10, 11, 12, 14.....	
J-15.....	280, 281		C. & O. loco- motive tests; no numbers given.	
J-16.....			Atlantic Coast Line loco- motive tests.	
J-17.....			Battleship Connecticut.	

^a Gas-producer test 188.^b Combustion test in kitchen stove made at Pittsburgh; see Bull. 14, Bureau of Mines, 1911, pp. 40-41.

LOCOMOTIVE-BOILER TESTS.

Two hundred tons of Arkansas semianthracite slack coal was made into briquets at St. Louis for the purpose of preliminary tests of their suitability for use in locomotives. Twelve tons of this coal was made on a Renfrow machine into 10-ounce biscuit-shaped briquets. The remainder was made into briquets $4\frac{1}{2}$ by $6\frac{1}{2}$ by $2\frac{1}{2}$ inches, averaging about $3\frac{1}{2}$ pounds each. From 6 to $6\frac{1}{2}$ per cent of pitch was used as a binding material in all these briquets.

After a few tests on a switching engine to determine the best method of firing these briquets, two tests were made on the regular passenger locomotive of the Missouri Pacific Railway from St. Louis to Sedalia, Mo., a distance of 188 miles. On a similar run made by this passenger train, with the same engine and fireman, lump coal was used from the

Consolidated Coal Co.'s No. 17 mine, at Collinsville, Ill., this coal being taken as a fair average of that ordinarily used by the Missouri Pacific Railway Co. The water and coal consumed were in each case carefully weighed, but in these preliminary tests no records were made of the steam pressures and temperatures of the feed water.

The engine steamed freely on each of the three tests. No clinkers were formed on the grates or tube sheet with the Illinois bituminous coal. A light clinker was formed over the grates in both runs with the Arkansas coal, which, however, did not interfere with the engine making the required amount of steam. The tube sheet clinkered on the run using Renfrow briquets, and it was necessary to clean off the clinker at Jefferson City, 125 miles from St. Louis. In both tests the briquets produced little smoke and were reported as being a satisfactory fuel for locomotives in the service of this railroad. The mechanical engineer of the company, under whose supervision these tests were made, expressed the opinion that the advantages gained in burning the briquets were more than sufficient to cover the cost of their manufacture.

In cooperation with the Missouri Pacific, the Lake Shore & Michigan Central, the Chicago, Rock Island & Pacific, the Chicago, Burlington & Quincy, and the Chicago & Eastern Illinois Railroads, 100 locomotive tests were made by the United States Geological Survey to determine the value, as a locomotive fuel, of briquets made from a large number of western coals. All tests were made on locomotives in actual service on the road. In some tests there was small opportunity for procuring elaborate data, but in others, where dynamometer cars were employed, it was possible to obtain more detailed results. The purpose that these tests were intended to serve was not so much to determine the evaporative efficiency of briquets as to investigate their behavior in practical use.

Briquets made from Arkansas semianthracite, two qualities of Indian Territory slack, Indian Territory screenings, Missouri slack, Indiana Brazil Block slack, coke breeze, and a mixture of coke breeze and washed Illinois coal were tested. Comparisons were drawn either with the same coal that was used in the briquet or with coal similar to it. In nearly every test the coal when burned in the form of briquets gave a higher evaporative efficiency than when burned in the natural state. For example, Indian Territory screenings give a boiler efficiency of 59 per cent, whereas briquets made from the same coal give an efficiency of 65 to 67 per cent. Decrease in smoke density, the elimination of objectionable clinkers, and a seeming decrease in the quantity of cinders and sparks are named as the chief reasons for this increased efficiency.

Locomotive tests at Altoona, Pa.—A series of steaming tests on a locomotive was made under the direction of A. W. Gibbs, general

superintendent of motive power of the Pennsylvania Railroad Co., by E. D. Nelson, engineer of tests at Altoona, Pa., to determine the comparative evaporative efficiency of briquets and of the run-of-mine coal of which they were made. The coal selected for the tests was taken from a mine working the Lower Kittanning coal bed near Lloydell, Cambria County, Pa. This coal is a very friable, bituminous coal, low in volatile matter. After exposure to the weather for 30 days before being briquetted the lot of coal showed little change except a decided increase in moisture, which, however, was eliminated in the briquetting process. The briquets tested on this locomotive were of two sizes, namely, those from the English machine and those from Renfrow No. 1 machine.^a The proportion of binding material present ranged from 5 to 8 per cent of water-gas pitch.

The locomotive used for all tests was a simple Atlantic (4-4-2) type of passenger locomotive of the Pennsylvania Railroad Co.'s class E-2a. The following conclusions are warranted from these tests according to W. F. M. Goss:

The evaporation per pound of fuel is greater for the briquetted Lloydell coal than for the same coal in its natural state. This advantage is maintained at all rates of evaporation. The capacity of the boiler is considerably increased by the use of briquetted coal. Briquetting appears to have little effect in reducing the quantity of cinders and sparks; the calorific value of cinders, however, is not so high in the briquetted as in the natural fuel. The density of the smoke with the briquetted coal is much less than with the natural coal. The percentage of binder in the briquet has little influence on smoke density. The percentage of binder for the range tested appears to have little or no influence on the evaporative efficiency.

The expense of briquetting under the conditions of the experiments adds about \$1 per ton to the price of the fuel, an amount that does not seem to be warranted by the resulting increase in evaporative efficiency. With careful firing, briquets can be used at terminals with a considerable decrease in smoke. The briquets appear to withstand exposure to the weather and suffer little deterioration from handling.

Tests of briquets on locomotive at Norfolk, Va.—In connection with the work at Norfolk, comparative steaming tests were made in stationary, marine, and locomotive boilers with run-of-mine coal and the same coal formed into briquets of two sizes. The tests in a locomotive boiler were undertaken by W. T. Ray and Henry Kreisinger to add cumulative evidence to work done at other places. They were made possible through the courtesy of the Seaboard Air Line Railway Co. in supplying both the locomotive and the coal used. During the trials the locomotive stood on a side track in the shop

^a For shapes and sizes of these briquets see the description of the Renfrow No. 1 machine (p. 26).

yards of the railway company at Portsmouth, Va. No running tests were made.

The primary object of the tests was to study the relative performances of the two types of briquets, and of the coal, with reference to efficiency, tendency to smoke, and the ease with which steam could be kept up, when each of the three varieties of fuel was burned at several rates of combustion.

All of the coal was in run-of-mine form from the Turkey Gap mine, Pocahontas No. 3 bed at Ennis, McDowell County, W. Va. Part of it went to the briquetting section of the fuel-testing plant at Norfolk, where it was designated Jamestown No. 13 and made into two sizes of briquets on the English and Renfrow No. 2 machines.*

Ray and Kreisinger^b give the following conclusions from this set of tests:

The combustion of suitable fuel can be kept at about the same completeness over a wide range. This is the result of the scrubbing action of rapidly moving currents of gases. In the case of combustion in the fuel bed, the CO, formed at the surface of the particles of fuel is replaced by fresh uncombined oxygen.

At low rates of working, run-of-mine coal gives a higher equivalent evaporation than briquets; at medium rates there is little difference; at high rates, briquets do considerably better. There is little difference between the large and the small briquets; the larger ones crumble less. The smaller briquets are easier to fire than are the larger ones; either form gives the fireman far less work and trouble than run-of-mine coal. In sparks briquet fires lose less than coal fires. On roads having heavy grades it will probably pay well to burn briquets, at least part of the time.

The high-capacity test run with briquets (test 14) was by no means the upper limit of fairly efficient combustion and evaporation. These particular briquets produced about as much smoke as the coal under similar conditions; some of the blame for this tendency to smoke may rest on the pitch binder.

Perhaps it would pay to add combustion chambers several feet long to the front of locomotive fire boxes, and to use a larger number of shorter and smaller boiler tubes.

The details of these tests may be found in Bureau of Mines Bulletin 34.^b

Tests on the Chesapeake & Ohio Railway.—In December, 1907, the United States Geological Survey cooperated with the Chesapeake & Ohio Railway Co. in making a series of comparative road tests of run-of-mine coal and briquets of the same coal on a locomotive in regular service. These tests were conducted under the supervision

* For sizes and shapes of these briquets see descriptions of these machines.

^b Ray, W. T., and Kreisinger, Henry, *Tests of run-of-mine and briquetted coal in a locomotive boiler*. 50 pp., 10 figs. (reprint of U. S. Geol. Survey Bull. 412).

of J. F. Walsh, superintendent of motive power. One car of fuel was used in tests on a passenger locomotive between Old Point and Richmond, Va., and four cars on through express locomotives between Washington, D. C., and Charlottesville, Va. The briquets were made at the fuel-testing plant of the Geological Survey at Norfolk, Va., from run-of-mine coal furnished by the railroad company, with 6 per cent of water-gas pitch used as a binder.

The coal for these tests came from the Fire Creek coal bed at Minden, New River district, West Virginia. The sample consisted of run-of-mine coal and reached the testing plant after having been exposed to the weather 8 to 16 days.

No attempt was made in these tests to make careful measurements of fuel and water; however, during the tests the following facts were developed: The briquets ignited freely, made an intensely hot fire, and emitted very little smoke. It was found that a comparatively heavy fire could be carried without danger of clinkering. Few ashes were left in the fire box or ash pan, and the cinder deposit was small. The results do not show that any apparent improvement in evaporative efficiency was obtained by the use of briquets, as compared with that obtainable from the natural fuel.

Tests on the Atlantic Coast Line Railroad.—W. F. M. Goss, in Bureau of Mines Bulletin 37^a gives a synopsis of tests of briquets on the Atlantic Coast Line Railroad. That synopsis is abstracted below.

The results of comparative tests of run-of-mine New River coal and of briquets of the same fuel, made in December, 1907, on a locomotive in the regular passenger service, are presented in the following table. The tests were conducted under the supervision of R. E. Smith, general superintendent of motive power of the railroad, in cooperation with the United States Geological Survey. Sixteen complete test trips were run between Rocky Mount and Wilmington, N. C., with the same engine, crew, and trains. An equal number of tests were made with run-of-mine coal furnished by the railroad company and with round and rectangular briquets made at the Geological Survey fuel-testing plant at Norfolk, Va., with 6 per cent of water-gas pitch binder.

Results of comparative tests of New River coal and of briquets.

Item.	Coal.	Briquets.
Number of test trips.....	16	16
Total pounds consumed.....	172,700	161,980
Average pounds consumed per trip.....	10,794	10,124
Average tons consumed per trip.....	5.397	5.062
Total engine-miles.....	1,984	1,984
Total car-miles.....	10,912	12,896
Pounds consumed per car-mile.....	15.8	12.5
Average cars per train.....	5.5	6.5

^a Goss, W. F. M., Comparative tests of run-of-mine and briquetted coal on locomotives. 58 pp., 4 pls., 25 figs. (reprint of U. S. Geol. Survey Bull. 363).

It is reported that from a practical standpoint the briquets thus tested were very satisfactory. Their use was found to eliminate all black smoke. No objectionable clinker was formed and the fuel seemed to burn completely.

TESTS ON TORPEDO BOAT "BIDDLE."

The fuel tests conducted at Norfolk included a detailed investigation of a number of Virginia and West Virginia coals that are bought by the United States Government for the Navy and for use in constructing the Panama Canal, and are extensively used by the merchant marine, manufacturing plants, and railroads. Through a cooperative arrangement with the Navy Department, steaming tests of the coals and their briquets were undertaken to determine their relative merits in marine boilers.

The tests were made on board the U. S. torpedo boat *Biddle*, beginning December 6, 1907, and ending January 27, 1908, by W. T. Ray and Henry Kreisinger. The results are reported in Bureau of Mines Bulletin 33.^a

The coal used in these tests came from the Sewell and Beckley beds in the New River district of West Virginia. With the particular equipment used in the test both coal and briquets were far from smokeless. The possibilities of coals of different composition are indicated by the data to be published later by the Bureau of Mines. A number of tests were made in which only little smoke was emitted while burning raw coal from the Pocahontas No. 3 coal bed, West Virginia, and briquets made therefrom. At a combustion rate of 120 pounds of the Pocahontas briquets per square foot of grate surface there was scarcely any smoke.

The coal used in these tests and in making the briquets was all in run-of-mine form and came from three different mines, all of which are located on the Chesapeake & Ohio Railway and in the New River coal field. These coals are designated Jamestown No. 6^b, Jamestown No. 9^b, and Jamestown No. 11^b. The briquets were made on the English and Renfrow No. 2 machines at Norfolk, Va.

The main object of these tests was to determine whether the use of briquetted coal on torpedo boats has any advantages over the use of raw coal. Besides comparing the economy obtained with briquetted and raw coal, the following properties of the two fuels were given particular attention: The tendency to smoke; the amount of sparks emitted from the stack; the rate at which steam can be made and the ease with which the fires are handled; and the ease of transferring fuel from the coal bunkers into the fireroom.

^a Ray, W. T., and Kreisinger, Henry, Comparative tests of run-of-mine and briquetted coal on the torpedo boat *Biddle*.

^b See table, p. 41.

In all, 21 tests were made, 10 with run-of-mine coal, 3 with English-machine briquets, and 8 with Renfrow-machine briquets. No test was run with the rate of combustion below 20 pounds per square foot of grate per hour. For the sake of comparison the tests of briquets were made at approximately the same rates of combustion as the tests of the run-of-mine coal of which the respective briquets were made.

Messrs. Ray and Kreisinger, who conducted these tests, give the following conclusions based on the results of the tests and applying only to the tests of New River run-of-mine coals when burned under a boiler of the Normand type and on vessels of the torpedo-boat class.

There is little or no gain in efficiency in burning briquets of either size. Both large and small briquets make as much (or more) smoke as run-of-mine coal. There seems to be more flaming in stack with briquets than with run-of-mine coal. About as many sparks are emitted from the stack whether briquetted or run-of-mine coal is burned. When burning briquets the fire need not be disturbed; with coal the fuel bed has to be broken up, generally after each firing. A somewhat higher boiler capacity can be obtained with briquets than with run-of-mine coal. Steam can be raised more quickly with briquets than with run-of-mine coal. Run-of-mine coal is transferred much more readily than briquets from the coal bunker to the fireroom. With briquets the capacity of a coal bunker is reduced 23 to 27 per cent.

The details of these tests may be found in Bureau of Mines Bulletin 33.^a

DOMESTIC-FURNACE TESTS.

Briquetted coal has been frequently tested in the hand-fired furnaces under the Heine boilers in the Survey fuel-testing plant at St. Louis. The results were so satisfactory that two series of tests were conducted to determine the value of this fuel for domestic-heating purposes. A number of evaporation tests were made on the house-heating boiler of the structural-materials laboratory, both briquets and coal being used. After these tests were well under way a carload of briquets was shipped to the University of Illinois engineering experiment station at Urbana, Ill., where two house-heating boilers were available. This equipment permitted more uniform conditions of pressure and capacity, making the results more valuable for comparison.

Perhaps the most important result obtained is that showing the relative value of different fuels for domestic purposes. The tests show that with a sectional boiler the effectiveness of different fuels

^a Ray, W. T., and Kreisinger, Henry, Comparative tests of run-of-mine and briquetted coal on the torpedo boat *Biddle*.

depends on the number of thermal units they contain. One basis for ascertaining the relative value of different fuels lies in the cost of evaporating 1,000 pounds of water. If, for example, at a certain place anthracite costs \$8 a ton and Pocahontas bituminous coal \$4, the cost of evaporating 1,000 pounds of water with anthracite would be 52 cents and with the bituminous coal 23.68 cents. This shows that a saving of about \$4 a ton for the same amount of work done would be effected by purchasing the bituminous coal.

A comparison of results with coal and with briquets shows no advantage in the briquets over coal of a size larger than screened nut. Briquetting good bituminous coal would be justified only when slack is used. The briquets tested gave much smoke, which was due to the pitch used as a binder.

The briquets and coal burned in the tests at St. Louis came from eleven States and Territories. There were 58 tests—11 on raw coals, 34 on round briquets, and 13 on square briquets. Most of the tests were run for about eight hours at a steam pressure of 2 to 3 pounds. The charge of fuel at each firing ranged from 55 to 175 pounds. The interval between firings varied considerably; in some tests coal was fired every half hour, and in others every two hours. The average efficiency in all the tests was 51.48 per cent; it ranged from 38.67 per cent with an Illinois coal to 65.36 per cent with a Virginia coal. The average percentage of builders' rated capacity developed was 59.2; it ranged from 44 per cent with an Illinois coal to 77.2 per cent with an Indian Territory coal. The lowest boiler horsepower developed was 12.1 and the highest 20.6. With the cost of fuel assumed at \$1 per 2,000 pounds, the cost of evaporating 1,000 pounds of water from and at 212° F. ranged from 5.56 cents for a briquetted Pennsylvania coal to 11.93 cents for a briquetted Illinois coal. Most of the briquets, whether made from eastern or western coal, smoked badly for several minutes after firing. Of the coals tested raw, six were western and five eastern. The high-volatile western coals smoked badly, but the eastern coals made comparatively little smoke.

An average capacity of 65 per cent was maintained on the 24 briquetting tests at Urbana, the range being from 53.2 to 71.4 per cent. To carry only 65 per cent of the rated radiating surface, the draft was wholly or very nearly cut off for one-half to three-fourths of the time. The boiler horsepower developed on boiler A ranged from 4.52 to 4.96 and on boiler B from 4.97 to 6.16. The average efficiency of the boiler and furnace (dry-coal basis) was 44.85 per cent for the 24 tests. A comparison of eight tests of briquets, of which four were made of large briquets and four of small briquets, shows that the large briquets invariably gave an appreciably higher efficiency, indicating that size is an important factor. With the cost of fuel assumed at \$1 per 2,000 pounds, the cost of evaporating 1,000 pounds of water

from and at 212° F. ranged from 6.74 cents with a Pennsylvania briquet to 12.47 cents with an Illinois briquet. The briquets started readily from a wood fire and burned well, but owing to the difficulty of obtaining complete combustion, the average efficiency from an eight-hour test was low. The formation of soot in the flues of boiler B was more troublesome and affected the economy more than in boiler A. So much soot was formed from partial combustion of the briquets in both boilers that the flues were blown after every test. In two tests the flues of boiler B were completely stopped at the end of the eight-hour run.

DISCUSSION OF RESULTS.

On comparing the results of these tests there seems to be no advantage in the briquets over coal of a suitable size for house-heating boilers. Briquetting a good bituminous coal would be justified only when slack is used and the gain from briquetting would lie almost entirely in the more favorable size of the fuel. This gain would be less for coals that coke readily than for noncoking coals. Briquets made from slack burn fairly well, as they allow the air to pass through the fuel bed.

The experiments showed that the pitch binders used are not suitable for a furnace working at the low temperatures common in a house-heating boiler, as they volatilized and generally escaped unburned or were deposited on the surface of the boiler. In the St. Louis tests this coating burned off once or twice a day, causing a high temperature in the flue and, as a consequence, danger from fire. There was a similar deposit of tarry matter on the boilers at Urbana, but it did not ignite, probably because of the better control of the fire. This deposit reduced the efficiency of the boiler. Briquets with binders that do not volatilize so readily would probably show superior results.

The briquets tested gave off much smoke, owing to the nature of the binder. The results of these tests indicate that coals containing the higher percentages of fixed carbon give the least smoke and the best results for economy.

SPECIAL TESTS.

Certain special combustion tests of briquets were made as described below.

· WATER-GAS MACHINE TEST.

The first special test comprised a water-gas machine test. The object of the test was (1) to determine whether briquets could be used as fuel for the production of water gas; (2) to compare the results obtained with those from retort coke; and (3) to observe the behavior of the briquets in the apparatus.

The briquets made in test 164 (Arkansas No. 13) were shipped to the Laclede Gas Co., St. Louis, Mo., and tested in an 8½-foot Lowe water-gas machine. This machine has a rated capacity of 1,000,000 cubic feet in 24 hours. The test was made under the direction of W. A. Baehr, chief engineer of the company.

Results of water-gas machine test of briquets.

Item.	Record.
Length of blast.....	minutes..... 5
Length of run.....	do..... 6
Total number of cycles.....	42
Total number of runs made.....	213
Total briquets used.....	pounds..... 85, 100
Total gas recorded by meter in 48 hours.....	cubic feet..... 1, 959, 000
Correction for relief holder.....	do..... 56, 100
Total gas made in 48 hours.....	do..... 1, 902, 900
Average temperature of gas metered.....	" F..... 73
Total gas made in 48 hours corrected to 60° F.....	cubic feet..... 1, 854, 000
Total oil used in 48 hours.....	gallons..... 7, 710
Average pressure of air during blast.....	inches of water..... 22.5
Average steam pressure during runs.....	pounds..... 29.2
Briquets per 1,000 feet of gas as metered (corrected to 60° F.).....	do..... 45.9
Five-hour coke per 1,000 feet of gas as metered (corrected to 60° F.).....	do..... 31.5
Oil per 1,000 feet gas as metered (corrected to 60° F.).....	gallons..... 4.16
Average luminosity of gas made.....	candlepower..... 19.4
Average amount of gas made:	
With briquets.....	do..... 4.65
With 5-hour coke.....	do..... 5.00
Average heat value of gas made:	
With briquets.....	B. t. u..... 624
With 5-hour coke.....	do..... 643
Average yield of gas as metered per run:	
With briquets.....	cubic feet..... 8, 940
With 5-hour coke.....	do..... 8, 450
Average amount of oil per run.....	gallons..... 36.2

CUPOLA TEST.

The briquets (coke breeze) made in test 247 (see pp. 264, 265) were sent to the Madison plant of the American Car & Foundry Co., of St. Louis, Mo., and were tested in a cupola. The object was to determine whether coke breeze when briquetted could be made to replace coke in foundry practice. It was hoped that by the addition of lime the briquets would be able to support the weight above until they became thoroughly coked.

Coke was used during the first charges and 150 pounds of briquets was used in each of the last five charges. The briquets were considerably broken during the charging of the iron and let the iron down below the melting zone.

COKING OF BRIQUETS AND BRIQUET MIXTURES.

The first experimental attempt to coke briquets was with some of the briquets made from Arkansas No. 4 coal and 6 per cent of pitch B. A box was filled with about 12 crushed briquets and then placed in one of the coke ovens, where it remained during a run of a coking coal. The result of this experiment was a mass of coke that was 12 to 15 inches long and somewhat metallic in appearance.

In the next experiment the box was filled with a mixture of Arkansas No. 4 coal with 6 per cent of pitch B, but without being passed through the press. The mixture was treated in the coke oven similarly to the other, but the results were not so satisfactory as in the first case. The next experiment was with the same mixture briquetted. The briquets were placed in a box and tested in the coke oven in the same manner as the others. The result of this test was masses of coke that retained the form of the original briquet. All of the coke obtained by these experiments, although not of the quality desired for iron smelting, could be used for lead and copper smelting.

Another experiment on a larger scale was with two tons of Arkansas No. 6 slack coal. This coal was mixed with 8 per cent of pitch A and then ground in the Williams mill and charged into a coke oven, together with an equal weight of the same mixture that had been previously briquetted. Both mixtures made coke, but not of the best quality. The experiments seemed to show that although a coke can be made by mixing coal and pitch a better quality is obtained by previously briquetting the mixture; however, sufficient experiments along this line have not yet been made to justify any definite conclusions.

DETAILS OF BRIQUETTING TESTS.

The data obtained from the general briquetting tests at the several experiment stations are presented below.

ALABAMA NO. 1.

Briquetting tests were made of a sample of bituminous coal (over 1-inch screen) from a mine $1\frac{1}{2}$ miles west of Horse Creek, Walker County, on the Frisco system. The tests were conducted at the St. Louis plant.

This is a coking coal which can be readily briquetted with hard pitch and, under extreme pressure, can be briquetted without the addition of any binder. There was 4.5 tons of this coal briquetted on the English machine with 7 per cent of the hard pitch A. The briquets were strong and rather satisfactory, except for porosity due to a lack of sufficient pressure. The briquets stand up well in the fire, but on long exposure to the weather disintegrate somewhat. The weight of the briquets averaged $5\frac{1}{2}$ pounds each.

Chemical analyses of coal and briquets.

	Car sample.	Briquets.
Laboratory No.....	1201	1194
Proximate:		
Air-drying loss..... per cent..	0.80	1.20
Moisture..... do.....	2.34	2.63
Volatile matter..... do.....	31.84	33.00
Fixed carbon..... do.....	53.28	50.96
Ash..... do.....	12.54	13.41
Sulphur..... do.....	.72	.94
Ultimate:		
Hydrogen..... do.....	4.86	4.71
Carbon..... do.....	73.30	72.65
Nitrogen..... do.....	1.69	1.42
Oxygen..... do.....	6.57	6.40
Heat value..... B. t. u..	12,856	12,735

ALABAMA NO. 2B.

Seven tons of bituminous coal from Carbon Hill, Walker County, on the Frisco system, was used in tests 131.1 to 131.4. Steaming test 410 was conducted with the briquets made from this fuel. The English briquets with 5 and 6 per cent of binder were good briquets; fracture clean; edges and surfaces firmer with 6 per cent than with 5 per cent binder. The Renfrow briquets made with 6 per cent of pitch showed evidence of shortage of binder; surfaces crumbled and fracture not clean. Seven per cent of binder made briquets with hard, firm surfaces; they broke without crumbling.

Briquetting tests.

	Test. No.			
	131.1.	131.2.	131.3.	131.4.
Size as shipped.....	r. o. m.	r. o. m.	r. o. m.	r. o. m.
Size as used:				
Over $\frac{1}{2}$ inch..... per cent..	2.8	2.8	2.8	2.8
$\frac{1}{2}$ inch to $\frac{3}{4}$ inch..... do.....	11.6	11.6	11.6	11.6
$\frac{3}{4}$ inch to $\frac{1}{2}$ inch..... do.....	21.2	21.2	21.2	21.2
$\frac{1}{2}$ inch to $\frac{1}{4}$ inch..... do.....	23.4	23.4	23.4	23.4
Through $\frac{1}{4}$ inch..... do.....	41.0	41.0	41.0	41.0
Details of manufacture:				
Machine used.....	English.	English.	Renfrow No. 1.	Renfrow No. 1.
Briquetting temperature..... °F..	175	175	149	149
Binder—				
Kind.....	W. G. P.	W. G. P.	W. G. P.	W. G. P.
Laboratory No.....	3410	3410	3410	3410
Amount..... per cent..	5	6	6	7
Weight of—				
Fuel briquetted..... pounds..	(e) 3.31	(e) 3.31	(e) 0.47	(e) 0.47
Briquet, average..... do.....				
Drop test (1-inch screen):				
Heid..... per cent..	91.5			
Passed..... do.....	8.5			
Weather test:				
Time exposed..... days..	197		201	200
Condition.....	B		B	B

e Total weight of fuel briquetted, 14,000 pounds.

Chemical analyses of coal and briquets.

	Car sample.	Test 131. ^a
Laboratory No.....	3211	
Proximate:		
Moisture..... per cent..	3.95	3.43
Volatile matter..... do.....	30.70	32.74
Fixed carbon..... do.....	50.76	51.34
Ash..... do.....	14.59	12.49
Sulphur..... do.....	1.12	1.24
Ultimate:		
Hydrogen..... do.....	4.30	4.77
Carbon..... do.....	68.94	72.17
Nitrogen..... do.....	1.55	1.58
Oxygen..... do.....	8.85	7.27

^a Composite sample of briquets from several lots.

Extraction analyses.

	Pitch.	Fuel.	Briquet.
		A-2B.	Test 131.
Laboratory No.....	3410	3211	
Air-drying loss..... per cent..		1.70	
Extracted by CS ₂ (as received)..... do.....	79.98	1.35	4.75
Pitch in briquet by analysis..... do.....			4.32
Heat value..... B. t. u.....	16,478	11,785	12,115

ALABAMA NO. 4.

Seven and one-half tons of bituminous coal from Cane Creek, 3 miles north of Belle Ellen, Bibb County, on the Louisville & Nashville Railroad, was used in briquetting tests 123A to 123H. Steaming test 413 was conducted with the briquetted fuel.

English briquets with 5.5 per cent of binder had edges that crumbled slightly, but the surfaces were firm; fracture slightly crumbly. With 6, 6.5, and 7 per cent binder the outer surfaces were smooth and hard, fracture clean, and broken surfaces rough but very firm.

Renfrow briquets made with 6 and 6.5 per cent binder crumbled easily, and broken surfaces were crumbly. With 7 and 7.5 per cent of binder tough briquets resulted, with hard and smooth outer surfaces; the briquets broke with clean fracture.

Briquetting tests.

	Test No.							
	123A.	123B.	123C.	123D.	123E.	123F.	123G.	123H.
Size as shipped.....	r. o. m.	r. o. m.	r. o. m.	r. o. m.	r. o. m.			
Size as used:								
Over $\frac{1}{8}$ inch, per cent.....	1.2							
$\frac{1}{8}$ inch to $\frac{1}{4}$ inch, per cent.....	4.5							
$\frac{1}{4}$ inch to $\frac{3}{8}$ inch, per cent.....	14.0							
$\frac{3}{8}$ inch to $\frac{1}{2}$ inch, per cent.....	23.5							
Through $\frac{1}{2}$ inch, per cent.....	56.8							
Details of manufacture:								
Machine used....	English.	English.	English.	English.	Renfrow No. 1.	Renfrow No. 1.	Renfrow No. 1.	Renfrow No. 1.
Briquetting temperature...°F..	179.6	179.6	179.6	179.6	149	149	149	149
Binder—								
Kind.....	W. G. P.	W. G. P.	W. G. P.	W. G. P.	W. G. P.	W. G. P.	W. G. P.	W. G. P.
Laboratory No.....	3410	3410	3410	3410	3410	3410	3410	3410
Amount, per cent.....	5.5	6	6.5	7.0	6	6.5	7.0	7.5
Average weight of briquet, pounds.....	3.37	3.37	3.37	3.37	0.43	0.43	0.43	0.43
Drop test (1-inch screen):								
Held... per cent.....		83.3						
Passed... do.....		16.7						
Weather test:								
Time exposed, days.....	214	214	214	214	214	214	214	214
Condition.....	B	B	B	B	C	C	C	B

Chemical analyses of coal and briquets.

	Car sample.	Test 123.
Laboratory No.....	3103	
Proximate:		
Moisture..... per cent.....	6.43	2.46
Volatile matter..... do.....	28.56	32.27
Fixed carbon..... do.....	52.09	56.55
Ash..... do.....	12.92	8.72
Sulphur..... do.....	1.06	1.18
Ultimate:		
Hydrogen..... do.....	4.83	4.72
Carbon..... do.....	73.82	76.96
Nitrogen..... do.....	1.26	1.32
Oxygen..... do.....	5.13	6.85

Extraction analyses.

	Pitch.	Fuel.	Briquets.
		Ala. No. 4.	Test 123.
Laboratory No.....	3410	3103	
Air-drying loss..... per cent.....		5.80	
Extracted by CS_2 (as received)..... do.....	79.98	.43	4.96
Pitch in briquet by analysis..... do.....			5.57
Heat value..... B. t. u.....	16,478	12,396	13,590

ARGENTINA NO. 1.

Coal received from the Province of Mendoza, Argentina, South America, was designated Argentina No. 1. Steaming test 485 was made of the briquetted fuel.

Renfrow briquets made with 7 per cent of pitch binder were very heavy and hard and gave a rough fracture without crumbling when broken. The coal particles were firmly cemented, and the briquets could be handled when warm without breaking. The sample of run-of-mine fuel used for the briquetting test was washed.

Chemical analyses of coal and briquets.

	Car sample.	Test 180.
Laboratory No.....	4079	
Proximate:		
Moisture.....per cent..	7.67	8.75
Volatile matter.....do..	18.39	21.45
Fixed carbon.....do..	31.09	33.75
Ash.....do..	42.85	36.06
Sulphur.....do..	1.21	.80
Ultimate:		
Hydrogen.....do..	2.76	3.41
Carbon.....do..	39.59	47.51
Nitrogen.....do..	1.03	.92
Oxygen.....do..	8.90	7.77

Extraction analyses.

	Pitch.	Fuel.	Briquets.
		Argentina No. 1.	Test 180.
Laboratory No.....	4543	4079	
Air-drying loss.....per cent..		3.0	4.30
Extracted by CS ₂ (as received).....do..	99.66	.06	6.10
Pitch in briquet by analysis.....do..			6.06
Heat value.....B. t. u..	16,969	6,320	7,515

Briquetting test.

	Test 180.		Test 180.
Size as shipped.....	r. o. m.	Drop test (1-inch screen):	
Size as used:		Held.....per cent..	84.5
Over $\frac{1}{2}$ inch.....per cent..	3.0	Passed.....do..	15.5
$\frac{1}{2}$ inch to $\frac{3}{4}$ inch.....do..	8.4	Tumbler test (1-inch screen):	
$\frac{3}{4}$ inch to 1 inch.....do..	15.0	Held.....do..	91.2
1 inch to $\frac{3}{4}$ inch.....do..	19.2	Passed.....do..	8.8
Through $\frac{3}{4}$ inch.....do..	54.4	Fines through 10-mesh sieve, per cent.....	94.0
Details of manufacture:		Weather test:	
Machine used.....	Renfrow No. 1	Time exposed.....days..	19
Briquetting temperature..°F.	185	Condition.....	A
Binder:		Absorption test:	
Kind.....	W. G. P.	Time immersed.....days..	19
Laboratory No.....	4543	Water absorbed.....per cent..	9.8
Amount.....per cent..	7	Average for first 4 days.....do..	1.44
Weight of.....		Specific gravity (apparent).....	1.467
Fuel briquetted..pounds..	6,000		
Briquet, average.....do..	0.603		

ARKANSAS NO. 1.

A sample of semibituminous coal (over 1½-inch screen) from Huntington, Sebastian County, on the Frisco system, was designated Arkansas No. 1, and briquetting tests of a portion of the sample were made at the St. Louis plant.

This coal was tested only with hard pitch A, as this was the only pitch available at the time the coal was briquetted. Six tons of coal was mixed with 9.25 per cent of this pitch and made into briquets on the English machine. The briquets were compact, but were too friable for handling, showing that they contained insufficient pitch. They were brownish and dirty. Judging from the results obtained in working with other Arkansas coals, this coal will require more than the usual proportion of pitch.

Although the actual steaming test of these briquets was satisfactory as compared with the tests of the coal, yet the briquets could not be considered a commercial product on account of their friability.

Chemical analyses of coal and briquets.

	Car sample.	Briquets.
Laboratory No.	1114	
Proximate:		
Air-drying loss..... per cent..	2.10	
Moisture..... do..	3.24	0.94
Volatile matter..... do..	17.48	21.21
Fixed carbon..... do..	66.69	67.65
Ash..... do..	12.61	10.20
Sulphur..... do..	1.24	1.73
Ultimate:		
Hydrogen..... do..	3.92	3.91
Carbon..... do..	76.57	79.76
Nitrogen..... do..	1.49	1.58
Oxygen..... do..	3.71	2.70
Heat value..... B. t. u..	13,129	13,707

ARKANSAS NO. 2.

A sample of semibituminous coal (over 1½-inch screen) from Bonanza, Sebastian County, on the Frisco system, was designated Arkansas No. 2. A part of the sample was subjected to briquetting tests at the St. Louis plant.

This coal was tested with the hard pitch X, which at the time was the only pitch available. The briquets were made in the English machine. Six tons of crushed lump coal was run through the machine, with approximately 11 per cent of pitch. The briquets were pitchy and wrinkled and probably contained nearer 15 per cent than 11 per cent of pitch. On account of the difficulty of setting the machine to feed accurate proportions of pitch and coal, the later experiments were conducted by weighing the desired quantities of binder. Pitch X, on account of its hardness, set too quickly in the molds, so that the briquets were insufficiently pressed and consequently porous.

Although they were not commercial briquets, a steaming test was made with them. The greater percentage of volatile matter in the briquets is due to the large proportion of pitch used in them.

Chemical analyses of coal and briquets.

	Car sample.	Briquets.
Laboratory No.....	1160	1112
Proximate:		
Air-drying loss per cent..	1.50	4.00
Moisture..... do.....	2.23	4.88
Volatile matter..... do.....	16.02	22.49
Fixed carbon..... do.....	72.55	60.30
Ash..... do.....	9.20	12.33
Sulphur..... do.....	1.87	1.32
Ultimate:		
Hydrogen..... do.....	4.08	4.06
Carbon..... do.....	80.63	76.65
Nitrogen..... do.....	1.41	1.54
Oxygen..... do.....	2.56	3.40
Heat value..... B. t. u..	13,750	12,915

ARKANSAS NO. 3.

A sample of semibituminous coal (over 1½-inch screen) from Jenny Lind, Sebastian County, on the Missouri Pacific Railroad, was designated Arkansas No. 3, and a part of the sample was subjected to briquetting tests at the St. Louis plant.

This coal was briquetted with 9.5 per cent of the hard pitch X. The coal was slightly moist, and passed through the machine without clogging the elevator. As the coal was friable it was reduced almost to a flour in the disintegrator. The briquets had smooth, polished surfaces. Some of the briquets as delivered by the machine showed cracks, perpendicular to the pressure, which may have been due to an excess of moisture. The briquets weighed on an average 6.8 pounds each.

The coal was also tested with the somewhat softer pitch A. One ton of the coal was mixed with 8.7 per cent of pitch. A large excess of steam was introduced into the pug mill so as to soften the pitch as much as possible. The briquets were smooth and easily handled, but pitchy. Under the same conditions of heat and pressure, 7.5 per cent of pitch A would have been sufficient, and if pitch of proper grade were used a still smaller amount would make a good commercial briquet. These briquets ignited readily and did not disintegrate in the fire.

Chemical analyses of coal and briquets.

	Car sample.	Briquets.
Laboratory No.....	1296	1293
Proximate:		
Air-drying loss..... per cent..	1.40	1.60
Moisture..... do.....	2.19	2.60
Volatile matter..... do.....	19.47	17.35
Fixed carbon..... do.....	66.71	62.04
Ash..... do.....	11.63	13.01
Sulphur..... do.....	1.23	1.41
Ultimate:		
Hydrogen..... do.....	4.02	3.46
Carbon..... do.....	77.00	72.47
Nitrogen..... do.....	1.56	1.42
Oxygen..... do.....	4.22	2.72
Heat value..... B. t. u.....	13,464	12,357

ARKANSAS NO. 4.

A sample of semibituminous slack coal from Denning, Franklin County, on the Missouri Pacific Railroad, was designated Arkansas No. 4. A part of the sample was subjected to briquetting tests at the St. Louis plant.

The sample consisted of slack coal, which was so saturated with water when received that it had to be dried before it could be used. When dried it was so brittle that the disintegrators of the English machine reduced it to flour, which choked the elevator and other parts of the machine. The first run made with this coal contained 12 per cent of hard pitch X, and although the briquets when first received from the machine were compact and smooth, they were unsatisfactory, and on exposure to the weather became friable and soon began to disintegrate. The next test of this finely divided coal was with pitch A. One ton was mixed with 10 per cent of pitch. The press was run faster, not only to increase the pressure, but also to give the hard pitch less time to set in the machine. At the same time the speed of the disintegrator was reduced in order not to pulverize the coal so completely. The briquets were pitchy, and, owing to the excess of water from condensation, many cracked when the pressure was relieved. Another ton of coal with 8 per cent of pitch A was used, but found to contain too much binder; still another ton was made into briquets with 6 per cent of pitch. The latter briquets were compact, but did not contain enough binder. Seven per cent of pitch A gave a better result than either 6 or 8 per cent.

Another run of this coal was made with 3 per cent of rosin and 2 per cent of pitch A. The briquets were clean and sharp, but were somewhat brittle and had a tendency to break into large fragments. They were, however, physically better than those with 6 or 8 per cent of pitch alone. In burning, they held together well but smoked more than those with pitch alone. No odor of rosin was given off, however. The same mixture was tested on the American machine, which

gave clean, polished briquets. They were stronger than those made on the English machine, indicating that rosin works to better advantage in the smaller and more rounded briquets. With pitch alone, however, and Arkansas No. 4 coal, a stronger briquet was made on the English machine than on the American. The weight of the briquets made with the rosin averaged 6.87 pounds each.

One ton of this coal was briquetted with 3 per cent of rosin and 3 per cent of pitch A. The mixture made excellent briquets, which were unusually clean and had sharp edges. They weighed 6.87 pounds each. This mixture was also tried on the American machine and gave exceedingly smooth and lustrous eggettes, which were stronger than the large briquets made on the English machine. Although no boiler test was made of the eggettes, some of them were thrown under the boiler, where they burned very readily without disintegration. They were tested in a cook stove, where they burned satisfactorily without any odor, but gave off considerably more smoke than either coal or briquets made from pitch alone as a binder. The specific gravity of these briquets was 1.13, whereas the specific gravity of the coal was 1.35. Their compression or crushing strength was 8,780 pounds, or 306 pounds per square inch.

The Arkansas No. 4 coal was further tested with the softer pitch B, which was one of the better pitches for briquetting. One ton was briquetted with 6 per cent of pitch. The briquets were clean and seemingly considerably stronger than the original lump coal and were capable of standing rough handling. They averaged 7 pounds each, and their specific gravity was 1.17. On the American machine the mixture gave polished eggettes, but these were not so strong as those made on the English machine.

Pitch B was then tried with rosin, the proportions being 95 per cent coal, 3 per cent rosin, and 2 per cent pitch. On account of the condensation of steam in the pug mill, there was considerable excess of water in the briquets, which, however, were smooth and sharp. The excess of moisture caused them to crack badly and they were difficult to handle while fresh. After cooling, they were still soft, and they were noticeably inferior to the briquets made with 3 per cent of rosin and 2 per cent of pitch A. They weighed 7.2 pounds each. The mixture was tested also on the American machine, which made harder eggettes than the corresponding briquets of the English machine. The smaller and rounder form is better adapted for the rosin binder and will result in a harder and tougher briquet.

Six tons of this coal was briquetted with 6 per cent of pitch B, in order to make a steaming test. Some of the briquets were made by running one side of the disintegrator of the English machine, and as they were received from the machine they were smooth and uniform, but soft. On cooling, the briquets became hard and tough.

Another ton was briquetted with both sides of the disintegrator running, which made smooth, uniform briquets, but it was almost impossible to handle them as they came from the machine. Therefore, only one side of the disintegrator was used, but the materials were run through the pug mill more rapidly and better results were obtained. All of these briquets were excellent in every respect. When these mixtures were tried on the American machine, the fine material made better eggettes than the corresponding briquets of the English machine, whereas the coarser material did not make as good eggettes as briquets. The weight of the briquets obtained with both sides of the disintegrator running was 6.92 pounds each; the coarser briquets weighed 6.56 pounds each. The specific gravity of the briquets was 1.17 and their crushing strength was 17,500 pounds, or 610 pounds per square inch.

Chemical analyses of coal and briquets.

	Raw fuel.	English briquets.	American briquets.
Laboratory No.		1320	1325
Proximate:			
Air-drying loss..... per cent.		2.10	0.10
Moisture..... do.	1.23	3.85	3.15
Volatile matter..... do.	12.52	14.06	13.60
Fixed carbon..... do.	73.69	71.95	66.03
Ash..... do.	12.21	10.11	12.19
Sulphur..... do.		1.65	1.70
Ultimate:			
Hydrogen..... do.		3.50	3.93
Carbon..... do.		80.69	78.31
Nitrogen..... do.		1.42	1.33
Oxygen..... do.		1.96	2.10
Heat value..... B. t. u.		13,414	13,464

ARKANSAS NO. 5.

A sample of semibituminous coal (half lump and half slack) from west of Coal Hill, Franklin County, on the Missouri Pacific Railroad, was designated Arkansas No. 5. A part of the sample was subjected to briquetting tests.

On account of the success obtained with the other Arkansas coals and the softer pitches, it was decided to make a sufficient quantity of briquets for a locomotive test on the Missouri Pacific Railway. For this purpose 17 tons of Arkansas No. 5 coal was briquetted with 6 per cent of pitch C. Excellent briquets were obtained, which were readily handled while warm, and on cooling stood a good deal of rough handling. They did not break so readily as the original lumps of coal. The briquets were tested on a locomotive of the Missouri Pacific Railway Co., and for comparison similar runs were made of an Illinois run-of-mine coal such as is commonly used on these locomotives. Three trips were made from St. Louis to Washington, Mo.,

and return, a total run of 324 miles for each fuel. The results of the three runs are given in the following table:

Locomotive tests of briquets of Arkansas No. 5 coal.

Briquets consumed.	Ash removed.	Front end cinders removed.	Total coal consumed.	Total ash removed.	Ash.	Miles to ton.
<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>	<i>Per cent.</i>	
9,720	1,260	194	29,594	4,615	15	21.89
10,049	1,535	182				
9,825	1,226	228				

Locomotive tests of run-of-mine Illinois coal.

Coal consumed.	Ash removed.	Front end cinders removed.	Total coal consumed.	Total ash removed.	Ash.	Miles to ton.
<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>	<i>Per cent.</i>	
11,265	1,980	130	34,015	5,695	16.4	19.05
11,050	1,568	112				
11,709	1,800	115				

The Arkansas briquets gave better results than the Illinois coal, indicating that by briquetting even an inferior coal it is possible to obtain a fuel that is superior not only to the original coal, but to good coals. L. Bartlett, one of the master mechanics of the Missouri Pacific Railway Co., under whose general direction these tests were conducted, says that "the briquets are too large for locomotive use. They must be thrown indiscriminately into the fire boxes, and if they take position side by side, the flame can not reach them. If, on the other hand, they lie in irregular positions, the spaces may be too large, resulting in a bad effect by allowing too much cold air to pass. In fact, the engine does not steam well until they are broken."

The briquets have always been considered too large, and in all steaming tests at the testing plant they were broken before the furnace was fed, and tests were made of the briquets to determine the amount of fine dust they made when broken.

Mr. Bartlett also noted the amount of smoke given off by this coal and stated that one advantage of the briquetted fuel was that it could be called a smokeless fuel as compared with the Illinois coal. This advantage would, of course, be considerable on switching engines in city freight yards.

The sulphur content of this coal, and consequently of the briquets, is rather high. The sulphur naturally attacked the grate bars, so that the briquets had a more injurious effect on the grate bars than coals containing less sulphur.

A further test of Arkansas No. 5 coal was made by mixing one-fourth of the soft pitch C and three-fourths of the hard pitch A, a

combination that made a good briquet, but not so good as that made with 6 per cent of pitch C. With more than three-fourths of hard pitch, it was necessary to use more steam for heating and greater pressure, and the resultant briquets were considerably inferior and more crumbly.

Chemical analysis.

	Car sample.		Car sample.
Laboratory No.....	1331	Ultimate:	
Proximate:		Hydrogen.....per cent..	3.65
Air-drying loss.....per cent..	1.10	Carbon.....do....	78.29
Moisture.....do....	2.38	Nitrogen.....do....	1.40
Volatile matter.....do....	12.68	Oxygen.....do....	2.25
Fixed carbon.....do....	72.88	Heat value.....B. t. u..	13,250
Ash.....do....	12.08		
Sulphur.....do....	1.99		

ARKANSAS NO. 6.

A sample of bituminous slack coal from Jenny Lind, Sebastian County, on the Missouri Pacific Railroad, was designated Arkansas No. 6. This coal was briquetted with 8 per cent of pitch A and made a briquet of fair quality. The briquets were made in order to test the coking value of the mixture, and the results are described under "Coking of Briquet Mixtures."

Proximate analysis.

	Car sample.		Car sample.
Laboratory No.....	1542	Proximate—Continued.	
Proximate:		Fixed carbon.....per cent..	68.50
Air-drying loss.....per cent..	3.00	Ash.....do....	13.81
Moisture.....do....	3.80	Sulphur.....do....	1.26
Volatile matter.....do....	13.89		

ARKANSAS NO. 7B.

Semibituminous slack coal from a mine 4 miles southwest of Midland, Sebastian County, on the Midland Valley Railroad, was designated Arkansas No. 7B. Briquets were made of 4,000 pounds of this washed slack coal.

English briquets made with 4, 5, and 6 per cent of binder were equally satisfactory, with hard, fine structure, glossy fracture, and well-molded, sharp edges.

Briquetting tests.

	Test 106.1.	Test 106.2.	Test 106.3.
Details in manufacture:			
Machine used.....	English.	English.	English.
Briquetting temperature.....°F..	179.6	179.6	179.6
Binder—			
Kind.....	C. T. P.	C. T. P.	C. T. P.
Laboratory No.....	2735	2735	2735
Amount.....per cent..	4	5	6
Drop test (1-inch screen):			
Held.....do.....	31.4		
Passed.....do.....	68.6		
Weather test:			
Time exposed.....days..	80	80	80
Condition.....	C	B	B

Chemical analysis.

	Car sam- ple.		Car sam- ple.
Laboratory No.....	2722	Ultimate:	
Proximate:		Hydrogen.....per cent..	3.94
Moisture.....per cent..	6.39	Carbon.....do.....	73.33
Volatile matter.....do.....	15.23	Nitrogen.....do.....	1.25
Fixed carbon.....do.....	62.88	Oxygen.....do.....	2.86
Ash.....do.....	15.00		
Sulphur.....do.....	2.24		

Extraction analyses.

	Pitch.	Fuel. Ark. No. 7B.
Laboratory No.....		2722
Air-drying loss.....per cent..		5.90
Extracted by CS_2 (as received).....do.....	58.56	
Heat value.....B. t. u.....		12,060

ARKANSAS NO. 13.

Bituminous slack from the mines at Denning on the St. Louis, Iron Mountain & Southern Railway was designated Arkansas No. 13. No steaming tests were made of this fuel either in the natural or briquetted form.

There was no noticeable difference in appearance between briquets made with 7 and 8 per cent of binder. Both were excellent, with hard, smooth surface and clean fracture, and were easily handled and piled while warm. Less binder could be used with increased pressure. The briquets with 8 per cent of binder made less slack in handling when cold. Those with 9 per cent were not satisfactory, as the pitch used had a much higher melting point and could not be successfully worked with the pressure available. Briquets from test 164 were used in the test of briquets in a water-gas machine. (See "Miscellaneous Tests of Briquets.")

Briquetting tests.

	Test 164.	Test 214.	Test 221.
Size as shipped.....	Slack.	Slack.	Slack.
Size as used:			
Over $\frac{1}{8}$ inch..... per cent..	1.4	1.4	1.4
$\frac{1}{8}$ inch to $\frac{1}{4}$ inch..... do.....	6.2	6.2	6.2
$\frac{1}{4}$ inch to $\frac{3}{8}$ inch..... do.....	10.6	10.6	10.6
$\frac{3}{8}$ inch to $\frac{1}{2}$ inch..... do.....	15.8	15.8	15.8
Through $\frac{1}{2}$ inch..... do.....	66.0	66.0	66.0
Details of manufacture:			
Machine used.....	Renfrow No. 1.	Renfrow No. 1.	Renfrow No. 1.
Briquetting temperature..... °F.	185	185	185
Binder—			
Kind.....	W. G. P.	W. G. P.	W. G. P.
Laboratory No.....	3885	3885	4635
Amount..... per cent..	7	8	9
Weight of—			
Fuel briquetted..... pounds..	140,000	40,000	5,000
Briquet, average..... do.....	0.451	0.464	0.489
Drop test (1-inch screen):			
Held..... per cent..	54.5	49.5	51.0
Passed..... do.....	45.5	50.5	49.0
Tumbler test (1-inch screen):			
Held..... do.....	79.0	81.5	86.0
Passed..... do.....	21.0	18.5	14.0
Fines through 10-mesh sieve..... do.....	87.7	87.3	88.9
Weather test:			
Time exposed..... days..	108		
Condition.....	B		
Absorption test:			
Time immersed..... days..	19	13	13
Water absorbed..... per cent..	17.2	13.6	10.2
Average for first 4 days..... do.....	2.50	2.36	1.72
Specific gravity (apparent).....	1.130	1.161	1.247

Chemical analyses of coal and briquets.

	Car sample.	Test 164.	Test 214.	Test 221.
Laboratory No.....	3798	4892		4626
Proximate:				
Moisture..... per cent..	2.91	1.49	1.05	1.78
Volatile matter..... do.....	12.65	15.11	16.50	15.98
Fixed carbon..... do.....	66.98	68.30	68.84	67.30
Ash..... do.....	17.51	15.10	13.61	14.96
Sulphur..... do.....	3.12	2.58	2.48	2.29
Ultimate:				
Hydrogen..... do.....	3.38	3.84	3.85	3.84
Carbon..... do.....	73.01	75.03	75.63	75.65
Nitrogen..... do.....	1.21	1.35	1.24	1.24
Oxygen..... do.....	1.15	1.88	1.70	1.71

Extraction analyses.

	Pitches.			Fuel.	Briquets.		
				Ark. No. 13.	Test 164.	Test 214.	Test 221.
Laboratory No.....	3885	4625	4683	3798	4892		4626
Air-drying loss..... per cent..				1.70	0.70		0.80
Extracted by CS ₂ (as received)..... per cent..	95.20	90.56	89.31	.09	5.76	6.29	6.29
Pitch in briquet by analysis..... per cent..					5.97	6.63	6.63
Heat value..... B. t. u.....	16,870	16,576	16,687	12,312	12,917	12,926	12,926

CALIFORNIA COAL—PITTSBURGH NO. 14.

A 3-car sample lot of run-of-mine lignite from the Ione mine, at Ione, Amador County, Cal., was designated Pittsburgh No. 14.

Briquetting tests 308, 309, 310, 311, 313, 314, 315, and 329 were made on it with the German lignite press. Gas-producer test 185 and house heating boiler test H90 were made on the raw lignite, and gas-producer test 188 and house-heating boiler tests H88 and H89 were made on the briquets.

The color of this lignite was light brown. It had a peculiar waxy feeling, like that of soapstone, and could be easily polished. A casual inspection showed that the lignite was rich in binding material. The results of subsequent tests and analyses confirmed this fact.

In the first test (308) of this lignite, the press operated much more easily than in the tests of Texas and North Dakota lignites. The dried material contained about 9.75 per cent moisture and the dies were set to give a high pressure. The briquets had rough surfaces, especially at the edges, but were of good shape. The cohesion test showed them to be stronger than the briquets made from any other lignite; in fact, they were surpassed in strength only by those made from the same lignite in test 329.

In test 309 the briquets were of excellent shape as they left the press, but on slight handling most of them broke in two along a shearing plane that extended diagonally from one long edge to another. Investigation indicated that this weakness resulted from the material being too dry to flow perfectly in the mold. In the next test (310) the moisture content of the briquet mixture was 1 per cent higher (8½ per cent) and the shearing was practically eliminated, although there were some signs of it. On account of this defect none of the briquets from tests 309 and 310 was saved.

Test 311 was made to see whether weathering injured the briquetting qualities of this lignite. The material used had been spread on the ground for 22 days and had become fine slack. In comparison with the car sample the only change shown by proximate analysis was a smaller percentage of moisture. Seemingly the weathering did not affect the briquetting qualities at all, for the test produced briquets having satisfactory shape, surface, and strength.

In test 313 the moisture in the briquet material was a little over 9 per cent, and a set of dies, ground to give a very heavy pressure, was used. The briquets were the weakest of any made from this lignite. Their lack of strength may have resulted in part from the dies being new, as dies work better after being polished by use.

Tests 314 and 315 were made to determine whether better briquets could be obtained by passing the undried material through a screen having $\frac{1}{8}$ -inch openings. The undried material for test 314 was not

screened, but that for test 315 was screened. With the same steam pressure in the drier, the dried material contained 14.6 per cent moisture in test 314 and only 10.13 per cent in test 315. The two lots of briquets had satisfactory strength, but both the drop and the tumbler tests showed that the screened material made the stronger briquets, the gain in strength being 12 per cent by the drop test and 26 per cent by the tumbler test. The two lots, however, did not differ much in general appearance, both being well formed and smooth.

Test 329 furnished the best lot of briquets made from this lignite. There was 10 per cent moisture in the briquet material; the pressure used was high, but lower than in tests 313, 314, and 315. The briquets had well-polished surfaces, and were the strongest made from any of the lignites named in this report. A "cohesive strength" of 71 per cent was shown by the drop test and 68 per cent by the tumbler test.

This lignite briquets very easily, as it contains more than enough natural binder. Excellent briquets were made with the German press by drying the ground lignite till it contained from $8\frac{1}{2}$ to 15 per cent moisture. The best briquets will be obtained when the material is ground so that all of it passes a $\frac{1}{4}$ -inch screen and 40 per cent passes a $\frac{1}{10}$ -inch screen, the briquets being subjected to a moderately high pressure. An extremely high pressure does not give the best results and increases the cost. The right pressure can be had by using a set of dies so ground that the tangent of the die angle is 6/89.

Summary of tests.

Test No.	Moisture in briquet material.	Pressure in mold.	Character of briquets.		Remarks.
			Form.	Strength.	
	<i>Per cent.</i>				
308.....	9.74	High.....	Good.....	Very good.....	Sides slightly cracked.
309.....	7.78	do.....	Excellent.....	Weak.....	Sheared diagonally.
310.....	8.51	do.....	Good.....	Good.....	No shearing evident.
311.....	7.68	do.....	do.....	do.....	Material too dry.
313.....	9.14	Very high.....	do.....	Fair.....	Do.
314.....	14.61	do.....	Very good.....	Very good.....	
315.....	10.13	do.....	do.....	do.....	
329.....	10.00	High.....	Excellent.....	Excellent.....	

Mixtures of this lignite with 25 and with 50 per cent of a Pittsburgh coal made excellent briquets. Patents have been granted for utilizing the natural binder of certain lignites to briquet mixtures of these lignites with bituminous-coal slack. Anthracite screenings can be used in place of the bituminous slack, and the substitution of such screenings would lessen the production of smoke.

Preliminary results from gas-producer tests of raw and briquetted samples of this lignite showed that the consumption of fuel, as fired per hour per brake horsepower developed, was 4.06 pounds for the

raw lignite and only 2.84 pounds for the briquets. The relative efficiency of the raw and the briquetted lignite as boiler fuel was roughly shown by house heating boiler tests H88, H89, and H90. Each pound of raw lignite evaporated 2.82 pounds of water at 212° F., and each pound of briquets evaporated 3.23 pounds.

The briquets resisted weathering well. Weathering does not entirely destroy the briquetting qualities of this lignite, as shown by the results of a test made on a sample that had been exposed to the weather for two years. It briquetted well, without addition of binder, under a pressure of only 6,000 pounds per square inch, after the sample had been dried and then heated by a steam jet for one minute.

Briquetting tests.

	Test No.							
	308.	309.	310.	311.	313.	314.	315.	329.
Size as shipped.....	r. o. m.	r. o. m.	r. o. m.	r. o. m.	r. o. m.	r. o. m.	r. o. m.	r. o. m.
Size as used:								
Over $\frac{1}{4}$ inch, per cent.....	2.0	3.5	2.0	2.0	6.5	10.0	5.0	3.0
$\frac{1}{4}$ to $\frac{1}{2}$ inch, per cent.....	18.5	21.0	18.5	18.5	29.0	33.0	29.0	29.0
$\frac{1}{2}$ to $\frac{3}{4}$ inch, per cent.....	31.0	33.0	29.0	32.5	23.0	21.5	23.5	26.0
$\frac{3}{4}$ to 1 inch, per cent.....	23.5	24.0	24.5	22.5	17.5	14.5	18.5	18.0
Through 1 inch, per cent.....	25.0	18.5	26.0	24.5	24.0	21.0	24.0	24.0
Details of manufacture:								
Machine used....	German.	German.	German.	German.	German.	German.	German.	German.
Briquetting temperature, F....	102	95	86	91	92	104	95	107
Binder used.....	None.	None.	None.	None.	None.	None.	None.	None.
Tangent of die, angle.....	6/89	6/89	6/89	6/89	7/89	7/89	7/89	6/89
Steam pressure on drier, pounds	10	15	13	10	10	5	5	7
Weight of—								
Fuel briquetted, pounds	6,875	1,790	2,457	3,407	2,915	13,937	5,304	9,000
Briquets, average, pounds.....	0.88	0.85	0.90	0.86		1.05	1.02	0.92
Moisture in briquet mixture, per cent.....	9.74	7.78	8.51	7.68	9.14	14.61	10.13	10
Drop test, 1-inch screen:								
Held..... per cent..	68	(*)		52	28.00	55	62	71
Passed..... do.....	32	(*)		48	72.00	50	38	29
Tumbler test, 1-inch screen:								
Held..... per cent..	68	(*)		48		38	64	68
Passed..... do.....	32	(*)		52		62	36	32
Fines through 10-mesh sieve, per cent.....	63.5	(*)		65.5		40	45.5	58.5
Weather test:								
Time exposed, days.....	66			66	66	66	66	66
Condition.....	A			A	B	A	A	A

* Briquets scrapped on account of diagonal cracks in them.

The tables following show the changes in composition of this fuel during storage at the plant for 68 days, until used for the last briquet-

ting test. The average of proximate analyses of 8 samples used for tests on different dates is also given. No mine sample was taken.

Chemical analyses.

	Car sample.	Sample as used on last test.	Average of 8 samples.
Date taken.....	Apr. 22	June 26	-----
Laboratory No.....	7621	8019	-----
Proximate:			
Air-drying loss..... per cent.	32.30	22.7	26.09
Moisture..... do.	39.46	27.97	34.52
Volatile matter..... do.	29.50	34.54	33.84
Fixed carbon..... do.	17.55	22.06	18.61
Ash..... do.	13.49	14.53	13.02
Sulphur..... do.	.97	1.58	1.08
Ultimate:			
Hydrogen..... do.	5.42		
Carbon..... do.	53.37		
Nitrogen..... do.	.61		
Oxygen..... do.	16.72		
Heat value..... B. t. u.	6,080	7,546	6,764

Proximate analyses of briquets.

	Test 308.	Test 309.	Test 310.	Test 311.	Test 313.	Test 314.	Test 315.	Test 329.
Laboratory No.....	7774	7757	7773	7775	7846	7861	7864	8018
Proximate:								
Moisture, per cent	9.07	6.66	7.85	7.34	9.17	12.06	10.40	9.82
Volatile matter, per cent.	46.53	43.99	47.53	49.04	46.28	44.95	46.60	46.48
Fixed carbon, per cent.	25.01	32.80	27.01	25.97	26.75	24.59	25.31	24.54
Ash..... per cent.	19.39	16.55	17.61	17.65	17.80	17.80	17.60	19.16
Sulphur, p. cent.	1.83	1.70	1.56	1.51	1.48	1.24	1.47	1.77

Extraction analyses.

Briquets.	Test 308.	Test 309.	Test 310.	Test 311.	Test 313.	Test 314.	Test 315.	Test 329.
Laboratory No.....	7774	7757	7773	7775	7846	7861	7864	8018
Extracted by CS_2 (moisture free), per cent.	7.02	7.01	7.60	7.76	7.98	8.35	7.97	-----
Heat value... B. t. u.	9,227	10,138	9,704	9,734	9,360	8,942	9,131	9,187

TESTS ON THE LADLEY PRESS.

Briquetting tests of this lignite were conducted with the Ladley press. This press is described on page 36, and shown in Plate XV. The lignite was dried in the Schulz drier at the Pittsburgh plant until it contained 6.79 per cent moisture and then shipped to Indianapolis.

METHOD OF HEATING LIGNITE.

In tests 1, 4, 5, 6, 7, 8, and 9 each pailful of the material was heated by dry steam from the nozzle of a hose, the steam being taken from the dome of the boiler. The steam escaping from the nozzle was slightly superheated, and although the loss of heat from radiation

was relatively large the lignite could be heated to a temperature of 208° F. The length of the heating was from one to two and a half minutes; the temperatures obtained were from 194° to 208° F. In tests 2, 3, and 10 the material was heated over a forge fire before pressing.

The molds for these tests were heated by placing a red-hot iron in a mold for some time. In some of the tests a gasoline torch, which gave a more even heat, was used. The dried and heated material was packed in the mold with a round bar and a sledge hammer, and the mold was again filled and tamped. Then the machine was started and the briquet was formed and ejected.

The briquets were 2½ inches in diameter and 3 inches long and weighed 10 to 10½ ounces.

The analyses of the briquet material before and after drying and of the briquets made from material heated by the two methods described are given in the following table:

Analyses of lignite and briquets.

		Material.		Briquets.	
		Before drying.	After drying.	Material heated by live steam.	Material heated by forge fire.
Moisture.....	per cent.	39.46	6.79	13.37	4.47
Volatile combustion.....	do.	29.50	48.72	44.36	47.43
Fixed carbon.....	do.	17.55	23.49	23.48	26.23
Ash.....	do.	13.49	21.00	18.79	21.87
Sulphur.....	do.	.97	1.86	1.54	1.82
Heat value.....	B. t. u.	6,080	9,126	8,662	9,538

The principal details of the tests are summarized as follows:

Details of tests 1 to 10 on the Ladley press.

Test No.	Heating of material.				Remarks.
	Method used.	Time of heating.	Temperature of material before pressing.	Quality of briquets obtained.	
1	Steam jet.....	<i>Minutes.</i>	<i>° F.</i>	Poor; too soft and dusty.	Mold too cold.
2	Forge fire.....	(a)	(b)	Poor; too dry.....	Water added before heating, to replace evaporation. Improvement made. Mold too cold. Satisfactory briquet.
3	do.....	(a)	199	Fair.....	
4	Steam jet.....	(a)	199	Good.....	
5	do.....	(a)	(b)	Fair.....	Do. Do. Do.
6	do.....	1½	208	Good.....	
7	do.....	2	205	Excellent.....	
8	do.....	2½	194	do.....	Do.
9	do.....	1	176	Good.....	Do.
10	Forge fire.....	(a)	212-230	Excellent.....	Do.

a Not noted. b Not taken. c A test, 7A, was made under same conditions as test 7 on the following day.

The briquets made in tests 7 and 10 were smooth and hard and their edges sharp and firm. The briquets from test 7 had fewer sur-

face cracks than those from test 10, although both lots were smooth enough. With steam in the steam jackets the molds would be hotter and the results might be even better.

The condensation of the steam used for heating the material in test 7 increased the moisture content of the material and of the resulting briquets. For this reason the briquets from that test had a lower heating value than those from test 10, in which the material was heated over a forge fire.

The tests demonstrated that without added binding material at least one American lignite could be made into good briquets by an American press.

STOVE AND GRATE COMBUSTION TESTS OF BRIQUETS.

Combustion tests of briquets of this California lignite were made at Pittsburgh, Pa., in a range and in an open heating grate to determine whether they fell to pieces while burning, the length of flame, and the quantity of smoke; also to determine the completeness of combustion as shown by the percentage of combustible matter in the refuse.

TESTS IN A KITCHEN RANGE.

The kitchen range had Dockash grates with openings one-half inch wide and could burn hard or soft coal. In the first trial the attempt was made to burn the briquets under the conditions present with the Pittsburgh coal regularly used. Some of the coal was mixed with the briquets, however, and the fire box was filled so high that proper combustion was impossible. For the next trial most of the coal was raked from the fire box and a fire gradually built with briquets alone. The results were excellent.

Briquets were added at half-hour intervals. The fire did not require much stirring, but it was raised slightly with a slice bar when it packed too much at the bottom. This did not cause excessive loss of unburned material through the grate, as was shown by the results of the test.

In five hours 32 briquets, or 32 pounds of fuel, was burned. The refuse weighed 8 pounds and was therefore 25 per cent of the fuel as fired. An analysis of the briquets showed the ash content to be 19.16 per cent.

Analysis of the refuse or "ashes" gave the following results:

	Per cent.
Moisture.....	2.52
Combustible matter.....	33.05
Ash (actual).....	64.43

Only one-third of the refuse was combustible matter, and, as the refuse was 25 per cent of the fuel fired, the loss of combustible through the grates was $33.05 \times 25 = 8.3$ per cent of the coal, or 11.5 per cent

of the combustible fired. The loss of 8.3 per cent is low for a kitchen stove, which at best does not burn fuel with high efficiency, and compares favorably with the percentage of loss for good bituminous coal burned in such a stove.

The proximate analyses of the briquets and of the refuse from the kitchen-stove and the heating-grate tests compare as follows:

Proximate analyses of briquets and of refuse from kitchen-stove and heating-grate tests.

Material.	Briquets.	Refuse from—	
		Kitchen-stove test.	Heating-grate test.
Moisture.....per cent..	9.82	2.52	0.77
Volatile matter.....do..	46.48		
Fixed carbon.....do..	24.54		
Ash.....do..	19.16	64.43	72.60
Sulphur.....do..	1.77		
Heat value.....B. t. u..	9,187		
Combustible matter.....per cent..	71.02	33.06	26.63

TESTS IN A HEATING GRATE.

Through the courtesy of the quartermaster's office, the author obtained the use of a heating grate in one of the vacant buildings on the arsenal grounds at Pittsburgh, Pa., for testing the combustion of the briquets in a heating grate. The grate was 24 inches wide, 9 inches deep at the center, and slightly less at the sides. The grate bars were of cast iron. The bottom bars ran from the back to the front and were spaced 1 inch apart. A fire was started in the grate with paper and a little wood, upon which was placed 25 pounds of briquets. A thermometer was hung on the outside of the building and another on the wall of the room about 4 feet in front and to one side of the fire and about 3 feet above the level of the grate. The temperatures taken were not expected to furnish data of interest. The main object of the tests was to note the combustion of the briquets and whether the economic results obtained were better than from tests under steam boilers.

Observations on grate test.

Time.	Temperature.		Weight of fuel as fired.
	Outside.	Inside.	
	° F.	° F.	Pounds.
9.20 a. m.....	37	51	25
9.45 a. m.....	37	55	20
10.30 a. m.....	36	67	20
12 noon.....	34	65	35
1.30 p. m.....	31	64	0
4 p. m.....	30	60	0
5 p. m.....	29	55	0

* Drop of temperature caused by opening an outside door.

No fuel was added to the fire after 12 noon. At 5 p. m., after burning five hours without attention, the fire was practically burned out. The table shows that in $7\frac{1}{2}$ hours 100 pounds of briquets was fired, giving 24 pounds, or 24 per cent, of refuse. After the briquets were well ignited they burned freely. They burned with a bright-yellow flame, much cleaner than the smoky flame from some bituminous coals, and the chimney showed only a slight yellowish smoke even directly after firing fresh fuel. The fire was unpleasantly hot to the face at a distance of 8 feet. No unpleasant odor could be detected in the room, though the gases from burning lignite have a characteristic odor and irritate the eyes and nose.

The combustible in the refuse was only 26.63 per cent, and as there was 24 per cent of refuse, the loss of fuel through the grate bars was approximately 6 per cent of the total fuel fired. This loss is no greater than from a coking coal burned under the same conditions. The loss through the grate of the kitchen range was approximately 8 per cent, or 2 per cent more than through the fireplace grate. The latter had wider spaces between the grate bars, but the fire in it did not need raking, whereas some raking of the kitchen-range grate was necessary.

The loss in both grates, however, was less than anticipated. The lignite used was noncoking, and one would naturally expect from it a larger loss of unburned material through the grate than from a coking coal.

CONCLUSIONS.

From the observed results of these combustion tests the following conclusions may be drawn:

The briquets ignite readily, make a hot fire, and burn freely until consumed.

Little shaking or poking of the fire is needed to obtain maximum efficiency from the fuel.

The loss of unburned fuel through the grates, 6 to 8 per cent, is not excessive and could be reduced by using step grates or grates with narrow spaces between the bars.

A grate measuring 8 by 24 inches is big enough for heating a large room with these briquets.

Under the test conditions little smoke was made; this smoke was light yellow, and would not be offensive in residential districts.

The briquets should prove a satisfactory domestic fuel, and, if they can be produced cheaply enough, should compete with other fuels for steam production.

COLORADO NO. 1.

A sample of run-of-mine subbituminous coal from Lafayette, Boulder County, on the Colorado & Southern Railroad of the Burlington system, was designated Colorado No. 1, and a part of the sample was subjected to briquetting tests.

It was first used undried with 10 per cent of pitch D. As these briquets were received from the English machine they were hard to handle, especially when there was an excess of steam. The pitch did not seem to adhere to the grains of lignite. On cooling, however, they became hard, black, and lustrous, but were too brittle. The adhesion of the pitch to the grains of lignite was not strong. The briquets seemed to be too pitchy, and if the adhesion between the pitch and the lignite were greater, 8 per cent of pitch D would undoubtedly be sufficient to give the desired results. Although there was no steaming test made with the briquets, they were burned under a boiler, and proved satisfactory. They weighed 5.98 pounds each. Some of the mixture was tested on the American machine, but the eggettes were much inferior to the briquets.

Chemical analysis.

	Car sample.		Car sample.
Laboratory No.....	1523	Ultimate:	
Proximate:		Hydrogen..... per cent..	4.91
Air-drying loss..... per cent..	6.00	Carbon..... do.....	70.66
Moisture..... do.....	18.68	Nitrogen..... do.....	1.41
Volatile matter..... do.....	34.88	Oxygen..... do.....	14.97
Fixed carbon..... do.....	40.45	Heat value..... B. t. u..	10,143
Ash..... do.....	5.99		
Sulphur..... do.....	.55		

ILLINOIS NO. 1.

A sample of bituminous coal (over 1-inch screen) from O'Fallon, St. Clair County, on the Belt Terminal Railroad of East St. Louis, was designated Illinois No. 1, and a part of the sample was subjected to briquetting tests at the St. Louis plant.

Briquets from this coal were first made with the hard pitch X as a binder. One and a quarter tons of this coal was briquetted on the English machine with 16 per cent of pitch X, but on account of the softness and roughness of the coal the briquets were brown and soft, resembling pressed damp loam. They were, however, compact.

Chemical analysis.

	Car sample.		Car sample.
Laboratory No.....	1261	Ultimate:	
Proximate:		Hydrogen..... per cent..	4.69
Air-drying loss..... per cent..	3.70	Carbon..... do.....	66.17
Moisture..... do.....	9.75	Nitrogen..... do.....	1.14
Volatile matter..... do.....	37.48	Oxygen..... do.....	8.83
Fixed carbon..... do.....	39.57	Heat value..... B. t. u..	11,025
Ash..... do.....	13.20		
Sulphur..... do.....	4.10		

ILLINOIS NO. 4.

A sample of bituminous coal (over 2-inch screen) from about 1 mile west of Troy, Madison County, on the St. Louis, Troy & Eastern Railroad, was designated Illinois No. 4 and a part of the sample was subjected to briquetting tests at the St. Louis plant.

Two tons of this coal was briquetted with 6 per cent of pitch D. The coal was not finely ground, only one side of the disintegrator being used. There was a great deal of water in the steam, and the briquets were full of the usual perpendicular cracks. A high pressure was used, and some of the briquets were larger than the normal size, a result that was due to the compression of the spring of the machine. The average weight of these briquets was 6.95 pounds. The briquets varied considerably in texture, some being hard and firm and others crumbly. They would not stand much rough handling. Their specific gravity was 1.16, and their crushing strength was only 5,100 pounds, or 181 pounds to the square inch. Some of the mixture was tried also on the American machine, and the egg-ettes were better than the briquets made on the English machine. They were lustrous and deep black.

One ton of this coal was briquetted with 8 per cent of pitch D, only one side of the disintegrator and the maximum pressure being used. Excellent briquets were obtained with this combination, but they were gray in color. They weighed 6.34 pounds each, and had a specific gravity of 1.11. The specific gravity of the coal is 1.34. The crushing strength of these briquets was 9,327 pounds or 325 pounds per square inch, as compared with 181 pounds to the square inch when 6 per cent of pitch D was used as a binder. The eggettes made of this mixture on the American machine were not so strong as the briquets.

In the next test with Illinois No. 4 coal, 2 tons of the coal was briquetted with 8 per cent of pitch E. The average weight of the briquets was only 6.15 pounds. Only one side of the disintegrator was used, and as the coal was hard the briquets were coarse and contained many noticeably large pieces of coal. The eggettes made from this mixture, on the other hand, were much better in quality

and rather lustrous, but of a decidedly brownish color. The specific gravity of the briquets was 1.13 and their crushing strength 7,265 pounds, or 263 pounds to the square inch. To test the effect of finer grinding, another ton of this coal was briquetted with 8 per cent of pitch E, both sides of the disintegrator being used. In these briquets there were a few grains of coal as big as a kernel of corn; but for the most part the material was too fine for the percentage of pitch, and the resultant briquets were crumbly, their crushing strength being only 209 pounds to the square inch, although they were well-pressed and lustrous and had a specific gravity of 1.03. There was little difference in the properties of the eggettes and the briquets made of this mixture. Another ton of this coal briquetted with 9 per cent of pitch E, both sides of the disintegrator and a maximum pressure being used, gave briquets of much better quality that would, however, have been improved by the addition of another one-half per cent of pitch. These briquets weighed 6.43 pounds each; their specific gravity was 1.08, and their crushing strength 9,125 pounds, or 352 pounds to the square inch. The eggettes made from this mixture were as good as the briquets and were more lustrous.

Another test was made with Illinois No. 4 coal and pitch H, which is a by-product from the manufacture of gas from heavy petroleum. Although this pitch would be rather soft to work on the English machine on a hot day, it is readily cracked and mixed on a cold day. One ton of the coal was briquetted with 8 per cent of pitch H, and the briquets were strong, clean, and satisfactory, being much better than any of the other briquets made with this coal. They were readily handled when taken from the machine, and when cold were hard and tough and capable of standing a great deal of rough handling. Their average weight was 6.81 pounds each and their specific gravity was 1.11. Their crushing strength was 12,810 pounds, or 446 pounds to the square inch, the briquets being hardest of all those made from Illinois coal.

The last test made with this coal was with a soft asphalt received from the Gulf Refining Co. In freezing weather this asphalt became hard enough to crack and could then be used on the English machine. On account of the cold weather there was considerable water in the steam, which produced cracks. Because of this fact and the low melting point and thinness of the asphalt while hot, the briquets were difficult to handle as they came from the machine. They were, however, well pressed and on cooling became tough and would stand much handling. The weight of these briquets was 6.77 pounds each. A better result can probably be obtained by using a small percentage of rosin with the asphalt.

Chemical analysis.

	Car sample.		Car sample.
Laboratory No	1417	Ultimate:	
Proximate:		Hydrogen	per cent. 4.59
Air-drying loss	per cent. 1.70	Carbon	do. 69.74
Moisture	do. 12.91	Nitrogen	do. 1.32
Volatile matter	do. 31.99	Oxygen	do. 9.46
Fixed carbon	do. 43.55	Heat value	B. t. u. 10,804
Ash	do. 11.64		
Sulphur	do. 1.32		

ILLINOIS NO. 7E.

Screened nut coal from a mine near Collinsville was designated Illinois No. 7E. Steaming test 516 was made of the unbriquetted fuel, but no steaming test was made of the briquets.

Renfrow briquets made with 8 per cent of binder were satisfactory, were easily transported from the machine without breaking, and became very hard when cold; their fractured surface was rough and their edges easily crumbled, owing to the excessive amount of clay present; the outer surfaces were smooth, with dull color.

Briquetting test.

	Test 244.		Test 244.
Size as used:		Drop test (1-inch screen):	
Over $\frac{1}{4}$ inch	per cent. 1.5	Held	per cent. 63.5
$\frac{1}{4}$ inch to $\frac{1}{2}$ inch	do. 6.6	Passed	do. 36.5
$\frac{1}{2}$ inch to $\frac{3}{4}$ inch	do. 14.2	Tumbler test (1-inch screen):	
$\frac{3}{4}$ inch to 1 inch	do. 22.0	Held	do. 87.5
Through 1 inch	do. 55.7	Passed	do. 12.5
Details of manufacture:		Fines through 10-mesh sieve,	
Machine used	Renfrow No. 1	per cent.	97.2
Briquetting temperature °F.	167	Absorption test:	
Binder—		Time immersed	days. 10
Kind	W. G. P.	Water absorbed	per cent. 7.4
Laboratory No.	4808	Average for first 4 days	do. 1.9
Amount	per cent. 8	Specific gravity (apparent)	1.257
Weight of—			
Fuel briquetted .. pounds.	12,000		
Briquets, average .. do.	0.472		

Chemical analyses of coal and briquets.

	Car sample.	Test 244.
Laboratory No.	4780	4928
Proximate:		
Moisture	per cent. 14.47	6.06
Volatile matter	do. 28.14	30.09
Fixed carbon	do. 35.16	40.34
Ash	do. 22.23	23.51
Sulphur	do. 3.73	5.09
Ultimate:		
Hydrogen	do. 3.84	2.57
Carbon	do. 56.08	59.48
Nitrogen	do. .91	.83
Oxygen	do. 8.82	5.67

Extraction analyses.

	Pitch.	Fuel.	Briquets.
		Ill. No. 7E.	Test 244.
Laboratory No.....	4906	4760	4928
Air-drying loss..... per cent..		10.00	1.70
Extracted by CS ₂ (as received)..... do.....	96.90	.42	8.09
Pitch in briquet by analysis..... do.....			7.96
Heat value..... B. t. u.....	16,864	8,797	10,021

ILLINOIS NO. 9C.

Bituminous slack coal from a mine near Staunton was designated Illinois No. 9C. Steaming tests 492 and 497 were made on this fuel in the briquetted form.

Briquets made on the English machine with 6 and 7 per cent of binder were satisfactory, having hard surfaces but rough fracture without crumbling.

Renfrow briquets made with 8 and 9 per cent of binder were inferior to those made on the English machine with 6 and 7 per cent of binder and were soft, although they stood handling from the machine; their fractured surfaces were crumbly, but the briquets were tough in texture. This coal requires high pressure to make good briquets.

From all these tests the briquets when cold and dried had excellent strength, as will be seen by noticing the results of the drop and the tumbler tests.

Briquetting tests.

	Test 189.	Test 190.	Test 223.	Test 234.
Size as shipped.....	Slack.	Slack.	Slack.	Slack.
Size as used:				
Over 1/2 inch..... per cent..	0.4	0.4	0.4	0.4
1/2 inch to 3/4 inch..... do.....	3.9	3.9	3.9	3.9
3/4 inch to 1 inch..... do.....	14.4	14.4	14.4	14.4
1 inch to 1 1/4 inch..... do.....	25.1	25.1	25.1	25.1
Through 1 1/4 inch..... do.....	56.2	56.2	56.2	56.2
Details of manufacture:				
Machine used.....	English.	English.	Renfrow No. 1	Renfrow No. 1
Briquetting temperature..... °F.	167	167	149	149
Binder—				
Kind.....	W. G. P.	W. G. P.	W. G. P.	W. G. P.
Laboratory No.....	4543	4543	4906	4906
Amount..... per cent..	6	7	8	9
Weight—				
Fuel briquetted..... pounds.	12,000	12,000	18,000	5,000
Briquets, average..... do.....	3.85	3.73	0.448	0.426
Drop test (1-inch screen):				
Heard..... per cent..	62.0	57.0	54.5	64.5
Passed..... do.....	38.0	43.0	45.5	35.5
Tumbler test (1-inch screen):				
Heard..... do.....	77.2	76.2	87.5	88.5
Passed..... do.....	22.8	23.8	12.5	11.5
Fines through 10-mesh sieve..... do.....	80.3	81.4	95.5	84.6
Weather test:				
Time exposed..... days.....	6	6		
Condition.....	B	B		
Absorption test:				
Time immersed..... days.....	19	19	13	13
Water absorbed..... per cent..	9.4	12.0	10.2	9.1
Average for first 4 days..... do.....	1.67	2.03	2.43	2.17
Specific gravity (apparent).....	1.146	1.124	1.098	1.128

Chemical analyses of coal and briquets.

	Car sample.	Test 189.	Test 190.	Test 234.
Laboratory No.....	4247	4406	4473	4874
Proximate:				
Moisture.....per cent.	15.25	14.62	17.43	5.43
Volatile matter.....do.	28.57	32.38	28.98	33.55
Fixed carbon.....do.	40.83	39.13	39.99	47.63
Ash.....do.	15.35	13.87	18.60	13.29
Sulphur.....do.	3.81	3.51	3.30	3.52
Ultimate:				
Hydrogen.....do.	4.17	4.33	4.31	4.44
Carbon.....do.	63.66	65.67	66.29	68.44
Nitrogen.....do.	.97	.88	.99	.94
Oxygen.....do.	8.59	8.77	7.94	8.30

Extraction analyses.

	Pitches.		Fuel.	Briquets.		
			Ill. No. 9.	Test 189.	Test 190.	Test 234.
Laboratory No.....	4543	4906	4247	4406	4473	4874
Air-drying loss.....per cent.			13.30	12.30	15.10	3.10
Extracted by CS_2 (as received).....do.	99.66	98.90	.63	6.80	6.18	9.41
Pitch in briquet by analysis.....do.				5.75	5.50	9.12
Heat value.....B. t. u.	16,909	16,864	9,790	10,276	9,943	11,713

ILLINOIS NO. 12B.

Bituminous coal from Bush, Williamson County, designated Illinois No. 12B, was No. 5 washed, and this sample was shipped uninspected and used washed in briquetting tests 166, 177, and 181.

Steaming test 463 was made of briquets from this sample.

English briquets made with 6.25 per cent of binder and Renfrow briquets made with 7 per cent of binder were of excellent quality. Briquets were easily handled when warm, although some difficulty was experienced in piling Renfrow briquets, as they stuck together on account of the low melting point of the pitch used. The surfaces of all briquets were hard and smooth, with rough, clean fracture and sharp edges.

Eight per cent of binder is more than is necessary to make satisfactory briquets from this fuel.

Briquetting tests.

	Test 166.	Test 177.	Test 181.
	No. 5.	No. 5.	No. 5.
Size as shipped.....			
Size used:			
Over $\frac{1}{4}$ inch..... per cent.	0.2	0.2	0.2
$\frac{1}{4}$ inch to $\frac{1}{2}$ inch..... do.	3.4	3.4	3.4
$\frac{1}{2}$ inch to $\frac{3}{4}$ inch..... do.	15.2	15.2	15.2
$\frac{3}{4}$ inch to 1 inch..... do.	24.4	24.4	24.4
Through 1 inch..... do.	56.8	56.8	56.8
Details of manufacture:			
Machine used.....	Renfrow No. 1.	English.	Renfrow No. 1.
Briquetting temperature..... °F.	131	185	131
Binder—			
Kind.....	W. G. P.	W. G. P.	W. G. P.
Laboratory No.....	4120	4543	4543
Amount..... per cent.	8	6.25	7
Weight—			
Fuel briquetted..... pounds.	100,000	51,000	53,000
Briquets, average..... do.	0.463	3.66	0.432
Drop test (1-inch screen):			
Held..... per cent.	66.5	85.1	65.5
Passed..... do.	33.5	14.9	34.5
Tumbler test (1-inch screen):			
Held..... do.	83.5	84.5	84.0
Passed..... do.	16.5	15.5	16.0
Fines through 10-mesh sieve..... do.	96.5	78.3	96.0
Weather test:			
Time exposed..... days.	63	12	19
Condition.....	B	A	B
Absorption test:			
Time immersed..... days.	19	19	19
Water absorbed..... per cent.	13.3	8.5	15.0
Average for first 4 days..... do.	2.15	1.00	2.13
Specific gravity (apparent).....	1.074	1.101	1.041

Chemical analyses of coal and briquets.

	Car sample.	Test 166.	Test 177.	Test 181.
Laboratory No.....	4201	4228		
Proximate:				
Moisture..... per cent.	15.87	6.99	5.87	5.78
Volatile matter..... do.	28.19	32.35	32.38	32.62
Fixed carbon..... do.	46.42	50.24	50.96	49.80
Ash..... do.	9.52	10.41	11.09	11.80
Sulphur..... do.	2.34	2.41	2.41	2.88
Ultimate:				
Hydrogen..... do.	4.56	4.49	4.20	4.59
Carbon..... do.	70.91	72.03	70.09	71.45
Nitrogen..... do.	1.26	1.09	1.11	1.09
Oxygen..... do.	9.17	8.61	10.28	7.57

Extraction analyses.

	Pitches.		Fuel.	Briquets.		
			III. No. 12B.	Test 166.	Test 177.	Test 181.
Laboratory No.....	4120	4543	4201	4228		
Air-drying loss..... per cent.			13.30	4.10		
Extracted by CS ₂ (as received)..... do.	97.70	99.66	.02	6.27	6.35	6.05
Pitch in briquet by analysis..... do.				6.40	6.35	6.05
Heat value..... B. t. u.	17,000	16,900	10,784	11,961	12,155	12,076

ILLINOIS NO. 20.

Bituminous coal from Staunton, Macoupin County, on the Litchfield & Madison Railroad, was designated Illinois No. 20. The sample consisted of screenings, and was shipped by the operator primarily for washing tests, but some of it was used in briquetting tests 103 and 104 in a mixture with Kentucky No. 2B (coke breeze).

Briquets made from coke breeze with 20 per cent, 33½ per cent, and 47 per cent of coal and 6 per cent of pitch were easily broken in handling. The coke breeze was washed, one-half being used wet and one-half put through a drier. Briquets made with dried fuel could be handled better when hot, but there was no difference between the two types when cold. Washed breeze was too high in ash for satisfactory fuel. These briquets were used in a preliminary test in switching locomotives. Briquets with 20 per cent of coal burned too slowly for switching service. Briquets with 33½ per cent of coal were satisfactory for switching, but burned too slowly for freight or passenger service. Briquets with 47 per cent of coal were satisfactory for all services. No analyses of the briquets were made.

Size as shipped, breeze through 1-inch coke fork and coal screenings; machine used, English; briquetting temperature, 179.6° F.; kind of binder, coal-tar pitch; proportion of binder, 6 per cent; laboratory number of binder, 2729; weight of fuel briquetted, 2,000 pounds.

Chemical analysis.

	Car sample.		Car sample.
Laboratory No.	2731	Ultimate:	
Proximate:		Hydrogen.....per cent..	4.43
Air-drying loss.....per cent..	12.40	Carbon.....do.....	64.71
Moisture.....do.....	14.68	Nitrogen.....do.....	1.17
Volatile matter.....do.....	31.32	Oxygen.....do.....	9.11
Fixed carbon.....do.....	40.32	Heat value.....B. t. u.....	10,063
Ash.....do.....	13.66		
Sulphur.....do.....	3.88		

ILLINOIS NO. 21.

Bituminous coal from Troy, Madison County, on the St. Louis, Troy & Eastern Railroad, was designated Illinois No. 21. Sample consisted of lump coal over a 2½-inch screen. A part of this sample, 14,000 pounds, was used in briquetting test 113. Steaming test 318 was made of the briquets from this test.

Machine used, English; briquetting temperature, 179.6° F.; kind of binder, coal-tar pitch; laboratory number of binder, 2933; amount of binder used, 5 and 6 per cent; weight of fuel briquetted, 14,000 pounds.

Briquets made with 5 per cent of binder were hard and firm, and stood rough handling. Weathering test: Days exposed, 75; condition of briquets made with 5 per cent binder, B.

Chemical analyses of coal and briquets.

	Car sample.	Test 113.
Laboratory No.....	2852	
Proximate:		
Air-drying loss.....per cent..	10.40	
Moisture.....do..	15.54	17.31
Volatile matter.....do..	31.26	30.40
Fixed carbon.....do..	42.27	42.28
Ash.....do..	10.93	10.01
Sulphur.....do..	1.38	1.28
Ultimate:		
Hydrogen.....do..	4.57	4.41
Carbon.....do..	68.69	70.01
Nitrogen.....do..	1.29	1.32
Oxygen.....do..	10.88	10.61
Heat value c.....B. t. u..	10,507	10,309

c Heat value of binder (Lab. No. 2933), 15,937 B. t. u.

ILLINOIS NO. 23B.

Bituminous slack coal, through 2-inch perforated screen, from Donkville, Madison County, on the St. Louis, Troy & Eastern Railroad, was designated Illinois No. 23B. Briquetting test 116 was made of this sample of coal. Steaming tests 321 and 322 were made of the briquetted fuel.

Briquets made with 5 and 6 per cent broke in handling. Those made with 6.5 per cent were satisfactory. In the weathering test all briquets were exposed 70 days; the condition of the briquets made with 5 per cent of binder was D; of those made with 6 per cent, C; and of those made with 6.5 per cent, B.

Briquetting test.

	Test 116.		Test 116.
Size as shipped.....	(a)	Details of manufacture—Continued.	
Details of manufacture:		Binder—Continued.	
Machine used.....	English.	Laboratory No.....	2933
Briquetting temperature, °F.....	179.6	Amount.....per cent..	5, 6, 6.5
Binder—		Weight of fuel briquetted.....pounds..	12,000
Kind.....	C. T. P.		

a Through 2½-inch screen.

Chemical analyses of coal and briquets.

	Car sample.	Test 116. ^a	Binder.
Laboratory No.....	2803		2933
Proximate:			
Air-drying loss.....per cent..	13.20		
Moisture.....do..	15.68	13.54	
Volatile matter.....do..	31.28	36.33	
Fixed carbon.....do..	37.45	41.67	
Ash.....do..	15.59	8.41	
Sulphur.....do..	3.98	3.21	
Ultimate:			
Hydrogen.....do..	4.36	4.86	
Carbon.....do..	62.72	69.30	
Nitrogen.....do..	1.07	1.14	
Oxygen.....do..	8.64	11.26	
Extract by CS ₂ (as received).....do..			64.20
Heat value.....B. t. u..	9,655	12,996	15,937

a Composite sample of briquets.

ILLINOIS NO. 28A.

Bituminous coal from bed No. 7 at Herrin, Williamson County, was designated Illinois No. 28A. This sample consisted of No. 5 washed coal, and a portion of it was used in briquetting tests 140, 141, 142, 143, 162, and 163 at St. Louis. Steaming test 459 was made with the briquets from tests 162 and 163 (equal weights of each).

There was no appreciable difference in briquets with 6, 7, and 8 per cent of binder made on the English machine and 8 per cent made on the Renfrow No. 1 machine. Renfrow briquets made with less than 8 per cent of binder were not satisfactory. The satisfactory briquets had hard, smooth outer surfaces, broke with clean, rough fracture without crumbling, and their edges were hard and sharp. Renfrow briquets could be handled warm and could be piled without crushing.

Briquetting tests.

	Test No.					
	140.	141.	142.	143.	162.	163.
Size as shipped.....	(a)	(a)	(a)	(a)	(a)	(a)
Size as used:						
Over $\frac{1}{4}$ inch.....per cent..	0.0	0.0	0.0	0.0	0.0	0.0
$\frac{1}{4}$ inch to $\frac{1}{2}$ inch.....do....	3.0	3.0	3.0	3.0	3.0	3.0
$\frac{1}{2}$ inch to $\frac{3}{4}$ inch.....do....	13.0	13.0	13.0	13.0	13.0	13.0
$\frac{3}{4}$ inch to 1 inch.....do....	20.2	20.2	20.2	20.2	20.2	20.2
Through 1 inch.....do....	63.8	63.8	63.8	63.8	63.8	63.8
Details of manufacture:						
Machine used.....	English.	English.	English.	Renfrow No. 1	Renfrow No. 1	English.
Briquetting temperature, °F.....	185	185	185	131	131	185
Binder—						
Kind.....	W.G.P.	W.G.P.	W.G.P.	W.G.P.	W.G.P.	W.G.P.
Laboratory No.....	3410	3410	3410	3410	3885	3885
Amount.....per cent.....	6	7	8	8	7	7
Weight of—						
Fuel briquetted...lbs.....	16,000	16,000	16,000	134,000	40,000	8,000
Briquets, average, pounds.....	3.44	3.54	3.45	0.502	0.468	3.62
Drop test (1-inch screen):						
Held.....per cent.....					61.5	
Passed.....do.....					38.5	
Tumbler test (1-inch screen):						
Held.....do.....					73.5	
Passed.....do.....					26.5	
Fines through 10-mesh sieve.....per cent.....					94.5	
Weather test:						
Time exposed.....days.....	182	182	182	173	127	173
Condition.....	A	B	A	B	B	B
Absorption test:						
Time immersed.....days.....					19	
Water absorbed, per cent.....					13.6	
Average for first 4 days, do.....					1.84	
Specific gravity (apparent).....					1.041	

^a Washed screenings.

Chemical analyses of coal and briquets.

	Car sample.	Test 140.	Test 142.	Test 143.	Tests 162 and 163.
Laboratory No.....	3409	-----	-----	4066	-----
Proximate:					
Moisture..... per cent..	14. 67	5. 81	5. 75	5. 78	7. 32
Volatile matter..... do..	26. 17	30. 31	31. 06	30. 58	30. 72
Fixed carbon..... do..	46. 62	49. 58	51. 50	51. 02	49. 84
Ash..... do..	12. 54	14. 30	12. 61	12. 62	12. 12
Sulphur..... do..	1. 05	1. 25	1. 21	. 95	1. 29
Ultimate:					
Hydrogen..... do..	4. 14	4. 21	4. 16	4. 35	4. 21
Carbon..... do..	70. 65	69. 69	71. 92	71. 89	71. 88
Nitrogen..... do..	1. 56	1. 36	1. 35	1. 36	1. 59
Oxygen..... do..	7. 72	8. 23	7. 95	8. 00	7. 85

Extraction analyses.

	Pitches.		Fuel.	Briquets.			
			Ill. No. 28A.	Test 140.	Test 142.	Test 143.	Tests 162 and 163.
Laboratory No.....	3410	3885	3409	-----	-----	4066	-----
Air-drying loss..... per cent..			9. 30	-----	-----		-----
Extracted by CS ₂ (as received), per cent.	79. 98	95. 20	0. 21	-----	-----	9. 15	-----
Pitch in briquet by analysis, per cent.				-----	-----	11. 21	-----
Heat value..... B. t. u..	16, 478	16, 870	10, 611	11, 561	11, 927	12, 026	11, 637

ILLINOIS NO. 28B.

Unwashed bituminous screenings from bed No. 7 at Herrin, Williamson County, was designated Illinois No. 28B, and a part of the sample was used in briquetting tests 144, 158, 159, 160, and 161. Steaming test 457 was made of the briquets from tests 159 and 161 (equal weights of each).

These briquets showed the same characteristics as those made from Illinois No. 28A coal, except that when cold their surfaces were tougher and less liable to break from abrasion than those of the briquets made of the washed fuel.

Briquetting tests.

	Test No.				
	144.	158.	159.	160.	161.
Size as shipped.....	(a)	(a)	(a)	(a)	(a)
Size as used:					
Over $\frac{1}{4}$ inch..... per cent.	0.4	0.4	0.4	0.4	0.4
$\frac{1}{4}$ inch to $\frac{1}{2}$ inch..... do.	3.8	3.8	3.8	3.8	3.8
$\frac{1}{2}$ inch to $\frac{3}{4}$ inch..... do.	13.4	13.4	13.4	13.4	13.4
$\frac{3}{4}$ inch to $\frac{7}{8}$ inch..... do.	24.8	24.8	24.8	24.8	24.8
Through $\frac{7}{8}$ inch..... do.	57.6	57.6	57.6	57.6	57.6
Details of manufacture:					
Machine used.....	Renfrow No. 1.	English.	English.	English.	Renfrow No. 1
Briquetting temperature..... °F.	131	185	185	185	131
Binder—					
Kind.....	W. G. P.	C. T. P.	C. T. P.	C. T. P.	W. G. P.
Laboratory No.....	3486	3692	3692	3692	3692
Amount..... per cent.	8	6	7	8	8
Weight of—					
Fuel briquetted..... pounds	123,000	20,000	28,000	20,000	30,000
Briquets, average..... do.	0.524	3.50	3.51	3.63	0.481
Drop test (1-inch screen):					
Held..... per cent.	49.0	77.9	72.3	80.6	41.0
Passed..... do.	51.0	22.1	27.7	19.4	59.0
Tumbler test (1-inch screen):					
Held..... do.	79.0	77.8	72.1	66.4	59.5
Passed..... do.	21.0	22.2	27.9	33.6	40.5
Fines through 10-mesh sieve..... do.	85.0	80.0	87.7	78.3	89.5
Weather test:					
Time exposed..... days	141	143	137	137	112
Condition.....	C	B	B	B	C
Absorption test:					
Time immersed..... days	19	19	19	19	19
Water absorbed..... per cent.	15.5	14.3	12.6	13.9	16.0
Average for first 4 days..... do.	1.98	1.40	1.30	1.40	2.14
Specific gravity (apparent).....	1.075	1.092	1.112	1.075	1.038

a Unwashed screenings.

b Equal weights of the two binders were used.

Chemical analyses of coal and briquets.

	Car sample.	Test 144.	Tests 158, 159, and 160.	Test 161.
Laboratory No.....	3423	4144	4150	4121
Proximate:				
Moisture..... per cent.	9.87	6.38	5.45	5.51
Volatile matter..... do.	29.21	31.07	31.65	30.86
Fixed carbon..... do.	48.47	51.19	51.74	52.41
Ash..... do.	12.45	11.36	11.16	10.92
Sulphur..... do.	2.19	1.55	2.04	2.03
Ultimate:				
Hydrogen..... do.	4.08	4.22	4.19	4.60
Carbon..... do.	70.23	71.82	71.62	71.92
Nitrogen..... do.	1.45	1.39	1.43	1.37
Oxygen..... do.	8.00	8.78	8.80	8.36

Extraction analyses.

	Pitches.				Fuel.	Briquets.		
					Ill. No. 28B.	Test 144.	Tests 158, 159, and 160.	Test 161.
Laboratory No.....	3486	3624	3692	3885	3423	4144	4150	4121
Air-drying loss..... per cent.					6.20			2.90
Extract by CS ₂ (as received)..... per cent.	87.57	99.60	69.71	95.20	.23	7.54	5.12	5.79
Pitch in briquet by analysis, per cent.						8.57	7.02	6.77
Heat value..... B. t. u.	16,407	16,193	16,103	16,870	11,329	12,094	12,161	12,082

ILLINOIS NOS. 29A AND 29B.

Bituminous screenings through a 2-inch shaking screen from bed No. 5 at Livingston, Madison County, were designated Illinois No. 29A, and briquetting tests 170 and 171 were made of a washed part of the sample. Steaming test 465 was made of the briquetted fuel.

Run-of-mine coal from the same location was designated Illinois No. 29B, and briquetting test 175 was made of a part of the sample. Steaming test 466 was made of the briquetted fuel.

Tests 170 and 171, made of washed screenings with 7 per cent of binder on the English and Renfrow No. 1 machines, respectively, furnished satisfactory briquets in each case. The briquets were very hard and tough and broke with rough fracture and hard edges. Renfrow briquets were well molded and stood handling when warm.

Briquets from test 175 made of the raw run-of-mine coal were identical with those made of the washed coal, except that the fracture was rougher and harder, as the fuel did not crush so fine in grinding.

Briquetting tests.

	Ill. No. 29A.		Ill. No. 29B.
	Test 170.	Test 171.	Test 175.
Size as shipped.....	(a)	(a)	r. o. m.
Size as used:			
Over $\frac{1}{2}$ inch..... per cent..	3.6	3.6	3.2
$\frac{1}{2}$ inch to $\frac{3}{4}$ inch..... do....	8.6	8.6	9.8
$\frac{3}{4}$ inch to 1 inch..... do....	16.2	16.2	16.4
1 inch to 1 $\frac{1}{2}$ inch..... do....	20.8	20.8	25.8
Through 1 $\frac{1}{2}$ inch..... do....	50.8	50.8	44.8
Details of manufacture:			
Machine used.....	English.	Renfrow No. 1.	Renfrow No. 1.
Briquetting temperature..... *F.	203	131	131
Binder—			
Kind.....	{ W. G. P. C. T. P. }	W. G. P.	W. G. P.
Laboratory No.....	{ 4319 4543 }	4543	4543
Amount..... per cent..	b 7	7	7
Weight of fuel briquetted..... pounds..	6,000	24,000	15,000
Average weight of briquet..... do....	3.84	0.437	0.465
Drop test (1-inch screen):			
Held..... per cent..	74.4	47.0	50.5
Passed..... do....	25.6	53.0	49.5
Tumbler test (1-inch screen):			
Held..... do....	94.0	75.0	77.5
Passed..... do....	6.0	25.0	22.5
Fines through 10-mesh sieve..... do....	1.2	87.3	90.2
Weather test:			
Time exposed..... days..	56	55	53
Condition.....	C	B	B
Absorption test:			
Time immersed..... days..	19	19	19
Water absorbed..... per cent..	17.0	21.5	18.8
Average for first 2 days..... do....	5.15	5.05	
Average for first 5 days..... do....			2.42
Specific gravity (apparent).....	1.053	0.969	1.052

* Washed screenings.

b Equal weights of the two pitches were used and the calorific value was determined from the separate calorific values of the two pitches.

Chemical analyses of coal and briquets.

	Car sample, Ill. 29A.	Car sample, Ill. 29B.	Test 170.	Test 171.	Test 175.
Laboratory No.	3963	3968	4233	4581	4245
Proximate:					
Moisture.....per cent..	13.10	12.47	14.23	12.29	10.00
Volatile matter.....do....	30.78	33.12	34.63	37.16	36.33
Fixed carbon.....do....	40.12	41.85	43.96	43.99	43.01
Ash.....do....	16.00	12.56	7.18	6.56	10.66
Sulphur.....do....	4.17	4.37	2.84	3.56	3.24
Ultimate:					
Hydrogen.....do....	4.37	4.55	4.78	4.91	4.82
Carbon.....do....	62.44	65.31	72.39	72.23	69.45
Nitrogen.....do....	1.02	1.14	1.06	1.07	1.26
Oxygen.....do....	8.96	19.66	10.07	10.26	9.03

Extraction analyses.

	Pitches.		Fuel.		Briquets.		
			Ill. No. 29A.	Ill. No. 29B.	Test 170.	Test 171.	Test 175.
Laboratory No.....	4319	4543	3963	3968	4233	4581	4245
Air-drying loss..... per cent.			8.60	7.30	11.80	10.40	7.80
Extract by CS ₂ (as received)..... do....	66.25	99.66	.76	.92	5.83	6.77	6.38
Pitch in briquet by analyses..... do....					6.17	6.09	5.53
Heat value..... B. t. u..	16,139	16,969	9,983	10,667	11,254	11,671	11,496

ILLINOIS NO. 30.

Bituminous nut coal, through a 3-inch and over a 2-inch shaking screen, from bed No. 7 at Shiloh Station, St. Clair County, on the Southern Railway, was used in briquetting tests 225, 226, and 228. Steaming test 511 was made of the briquets.

Briquets from all three tests were satisfactory and had hard, rough exteriors, and broke with rough, uneven fractures, but had somewhat crumbly edges. All briquets had a bluish color and fracture, which showed an excess of pitch. The Renfrow briquets showed the results of insufficient pressure. No analyses were made of the briquets resulting from test 228.

Briquetting tests.

	Test 225.	Test 226.	Test 228.
Size as shipped.....	(a)	(a)	(a)
Size as used:			
Over $\frac{1}{2}$ inch..... per cent..	2.0	2.0	2.0
$\frac{1}{2}$ inch to $\frac{3}{4}$ inch..... do....	7.0	7.0	7.0
$\frac{3}{4}$ inch to $\frac{7}{8}$ inch..... do....	17.4	17.4	17.4
$\frac{7}{8}$ inch to $\frac{15}{16}$ inch..... do....	20.8	20.8	20.8
Through $\frac{15}{16}$ inch..... do....	52.8	52.8	52.8
Details of manufacture:			
Machine used.....	English.	English.	Renfrow No. 1.
Briquetting temperature..... °F.	185	185	188
Binder—			
Kind.....	W. G. P.	W. G. P.	W. G. P.
Laboratory No.....	4806	4806	4806
Amount..... per cent..	6	7	8.5
Weight of:			
Fuel briquetted..... pounds..	6,000	6,000	6,500
Briquets, average..... do....	3.50	3.45	0.463
Drop test (1-inch screen):			
Held..... per cent..	80.2	81.5	47.0
Passed..... do....	19.8	18.5	53.0
Tumbler test (1-inch screen):			
Held..... do....	75.9	79.0	82.5
Passed..... do....	24.1	21.0	17.5
Fines through 10-mesh sieve..... do....	82.6	85.0	96.0
Absorption test:			
Time immersed..... days..	13	13	23
Water absorbed..... per cent..	8.6	8.5	16.2
Average for first 4 days..... do....	1.35	1.98	2.10
Specific gravity (apparent).....	1.087	1.080	1.080

* Nut, washed.

Chemical analyses of coal and briquets.

	Car sample.	Tests 225 and 226.
Laboratory No.....	4364	4714
Proximate:		
Moisture..... per cent..	11.69	9.75
Volatile matter..... do....	35.70	39.25
Fixed carbon..... do....	39.42	42.69
Ash..... do....	13.19	8.31
Sulphur..... do....	4.38	3.21
Ultimate:		
Hydrogen..... do....	4.71	4.80
Carbon..... do....	64.72	71.70
Nitrogen..... do....	1.06	1.13
Oxygen..... do....	9.61	9.60

Extraction analyses.

	Pitch.	Fuel.	Briquets.
		Ill. No. 30B.	Tests 225 and 226.
Laboratory No.....	4806	4364	4714
Air-drying loss..... per cent..		8.70	5.3
Extracted by CS ₂ (as received)..... do....	96.90	.61	6.18
Pitch in briquet by analysis..... do....			5.80
Heat value..... B. t. u..	16,864	10,699	11,977

ILLINOIS NO. 31.

Bituminous screenings, over a 1½-inch shaking screen, from bed No. 6, at Worden, St. Clair County, on the Wabash Railroad, were designated Illinois No. 31, and a part of this sample was used in briquetting tests 185, 186, 224, and 237. Steaming tests 489 and 491 were made of the briquets.

The briquets were porous, with soft surfaces, rough fracture, and easily crumbled edges. English briquets were handled from the machine without breaking. Renfrow briquets were not satisfactory, owing to low pressure. This coal, although dirty, should be a good briquetting coal.

Briquetting tests.

	Test 185.	Test 186.	Test 224.	Test 237.
Size as shipped.....	(a)	(a)	(a)	(a)
Size as used:				
Over ½ inch..... per cent..	3.0	3.0	3.0	3.0
½ inch to ¾ inch..... do....	8.5	8.5	8.5	8.5
¾ inch to 1 inch..... do....	17.0	17.0	17.0	17.0
1 inch to 1½ inch..... do....	24.0	24.0	24.0	24.0
Through 1½ inch..... do....	47.5	47.5	47.5	47.5
Details of manufacture:				
Machine used.....	Renfrow No. 1	English.	English.	Renfrow No. 1
Briquetting temperature, °F.....	140	176	149	149
Binder—				
Kind.....	W. G. P.	W. G. P.	W. G. P.	W. G. P.
Laboratory No.....	4543	4543	4683	4906
Amount..... per cent..	7	7	7	8
Weight of—				
Fuel briquetted..... pounds..	12,000	12,000	11,000	10,000
Briquets, average..... do....	0.460	3.77	3.91	0.442
Drop test (1-inch screen):				
Held..... per cent..	64.5	79.7	24.2	34.5
Passed..... do....	35.5	20.3	75.8	65.5
Tumbler test (1-inch screen):				
Held..... do....	91.5	78.8	43.3	80.5
Passed..... do....	8.5	21.2	56.7	19.5
Fines through 10-mesh sieve..... do....	94.1	75.7	65.0	87.5
Weather test:				
Time exposed..... days..	11	10		
Condition.....	B	A		
Absorption test:				
Time immersed..... days..	19	19	13	13
Water absorbed..... per cent..	14.5	9.2	7.4	13.1
Average for first 4 days..... do....	2.70	1.60	1.67	1.93
Specific gravity (apparent).....	1.053	1.144	1.167	1.123

a Screenings.

Chemical analyses of coal and briquets.

	Car sample.	Test 185.	Test 186.	Test 224.	Test 237.
Laboratory No.....	4376	4397	4382	4830	4890
Proximate:					
Moisture..... per cent..	13.10	10.81	14.52	9.17	6.96
Volatile matter..... do....	32.16	33.72	33.74	33.03	35.21
Fixed carbon..... do....	41.49	42.74	39.40	43.90	42.85
Ash..... do....	13.25	12.73	12.34	13.90	14.96
Sulphur..... do....	3.66	3.50	3.36	3.90	3.78
Ultimate:					
Hydrogen..... do....	4.58	4.54	4.68	4.33	3.57
Carbon..... do....	65.95	67.31	67.91	66.14	59.48
Nitrogen..... do....	1.19	1.04	1.12	1.01	0.83
Oxygen..... do....	8.82	8.92	7.92	8.93	5.67

Extraction analyses.

	Pitches.			Fuel.	Briquets.			
				Ill. No. 31.	Test 185.	Test 186.	Test 224.	Test 237.
Laboratory No.....	4543	4683	4806	4376	4397	4385	4830	4880
Air-drying loss..... per cent..				9.70	8.00	11.10	4.70	2.80
Extract by CS ₂ (as received)..... per cent..	99.66	89.31	96.90	.44	6.50	6.21	6.72	7.58
Pitch in briquet by analysis..... per cent..					6.11	5.86	7.08	7.41
Heat value..... B. t. u..	16,969	16,637	16,864	10,393	11,002	10,517	10,921	11,158

ILLINOIS NO. 33.

Bituminous-coal screenings, through 1½-inch shaking screen, from bed No. 7 at Trenton, Clinton County, on the Baltimore & Ohio Railroad, were designated Illinois No. 33, and a part of the sample was used in briquetting tests 206, 207, 210, and 235. Steaming test 513 was made of the briquets.

The surfaces of all briquets were soft, their fracture ragged, and their edges easily broken. Greater percentages of binder had no apparent effect on their appearance. Increased moisture content made a very soft, warm briquet, which hardened on cooling with a harder surface than those with less moisture.

Briquetting tests.

	Test 206.	Test 207.	Test 210.	Test 235.
Size as shipped.....	Screenings.	Screenings.	Screenings.	Screenings.
Size as used:				
Over ½ inch..... per cent..	2.0	2.0	2.0	2.0
½ to ¾ inch..... do.....	9.8	9.8	9.8	9.8
¾ to 1 inch..... do.....	26.4	26.4	26.4	26.4
1 to 1½ inch..... do.....	27.0	27.0	27.0	27.0
Through 1½ inch..... do.....	34.8	34.8	34.8	34.8
Details of manufacture:				
Machine used.....	English.	English.	Renfrow No. 1.	Renfrow No. 1.
Briquetting temperature..... ° F..	149	149	158	167
Binder—				
Kind.....	W. G. P.	W. G. P.	W. G. P.	W. G. P.
Laboratory No.....	4683	4683	4683	4806
Amount..... per cent..	7	6	7	8
Weight of—				
Fuel briquetted..... pounds..	8,000	7,000	6,000	15,000
Briquets, average..... do.....	4.07	4.12	0.474	0.484
Drop test (1-inch screen):				
Held..... per cent..	50.0	61.8	74.5	34.5
Passed..... do.....	50.0	38.2	25.5	65.5
Tumbler test (1-inch screen):				
Held..... do.....	55.6	60.3	94.0	89.5
Passed..... do.....	44.4	39.7	6.0	10.5
Fines through 10-mesh sieve..... do.....	66.6	72.1	90.8	94.7
Absorption test:				
Time immersed..... days..	10	10	10	10
Water absorbed..... per cent..	7.6	9.4	11.1	6.5
Average for first 4 days..... do.....	2.30	2.40	2.98	1.70
Specific gravity (apparent).....	1.291	1.280	1.232	1.298

Chemical analyses of coal and briquets.

	Mine sample.	Tests 206 and 207.	Test 235.
Laboratory No.....	4385	4728	4888
Proximate:			
Moisture..... per cent..	15.06	11.50	9.47
Volatile matter..... do.....	29.48	25.44	23.25
Fixed carbon..... do.....	45.81	42.37	40.75
Ash..... do.....	9.65	20.69	21.53
Sulphur..... do.....	1.05	1.52	1.56
Ultimate:			
Hydrogen..... do.....		3.72	3.93
Carbon..... do.....		61.46	60.07
Nitrogen..... do.....		1.02	1.03
Oxygen..... do.....		8.70	9.47

Extraction analyses.

	Pitches.		Fuel.	Briquets.	
			Ill. No. 33.	Tests 206 and 207.	Test 235.
Laboratory No.....	4683	4806	4385	4728	4888
Air-drying loss..... per cent..			4.80	5.50	3.90
Extract by CS ₂ (as received)..... do.....	89.31	96.90	.28	4.93	7.42
Pitch in briquet by analysis..... do.....				5.23	7.40
Heat value..... B. t. u..	16,637	16,864	10,726	9,823	9,761

INDIANA NO. 1.

A sample of bituminous run-of-mine coal from Mildred, Sullivan County, on the Evansville & Terre Haute Railroad, was designated Indiana No. 1, and a part of the sample was subjected to briquetting tests at the St. Louis plant.

Four tons of this coal was briquetted in the English press with 7 per cent of pitch D. The coal was first washed and then dried. The briquets were brown in color, readily handled as they came from the machine, and when cold stood rough handling. The average weight of the briquets was 6.8 pounds each. Eggettes made of the mixture on the American machine were better and stronger than the briquets.

Chemical analyses of coal and briquets.

	Car sample.	Briquets.
Laboratory No.....	1507	1521
Proximate:		
Air-drying loss..... per cent..	3.00	4.90
Moisture..... do.....	11.40	11.74
Volatile matter..... do.....	33.81	38.79
Fixed carbon..... do.....	41.39	43.23
Ash..... do.....	13.40	8.24
Sulphur..... do.....	2.50	2.03
Ultimate:		
Hydrogen..... do.....	4.63	4.61
Carbon..... do.....	68.11	75.77
Nitrogen..... do.....	1.33	1.47
Oxygen..... do.....	7.99	8.46
Heat value..... B. t. u..	11,061	12,087

INDIANA NO. 1B.

Bituminous screenings (through 1-inch screen) from Mildred, Sullivan County, on the Evansville & Terre Haute Railroad, were designated Indiana No. 1B. The sample was used in briquetting test 217. No steaming test was made of the briquets.

Briquets made with 8.5 per cent of binder on the Renfrow No. 1 machine were satisfactory when the fuel was very dry; excess of moisture made briquets that were soft when warm and crumbled easily when cold. The briquets had a porous, dull surface and a rough fracture with ragged edges, but were tough and hard, producing only a small amount of slack in handling and showing no evidence of sticking together when piled warm. Locomotive and house boiler tests were made of briquets from test 217.

Briquetting test.

	Test 217.		Test 217.
Size as shipped.....	Screenings.	Drop test (1-inch screen):	
Size as used:		Held..... per cent..	54.5
Over $\frac{1}{2}$ inch..... per cent..	5.0	Passed..... do.....	45.5
$\frac{1}{2}$ inch to $\frac{3}{4}$ inch..... do.....	12.4	Tumbler test (1-inch screen):	
$\frac{3}{4}$ inch to $\frac{1}{2}$ inch..... do.....	22.4	Held..... do.....	86.5
$\frac{1}{2}$ inch to $\frac{3}{4}$ inch..... do.....	23.4	Passed..... do.....	13.5
Through $\frac{3}{4}$ inch..... do.....	36.8	Fines through 10-mesh sieve,	
Details of manufacture:		per cent.....	95.0
Machine used.....	Renfrow No. 1.	Absorption test:	
Briquetting temperature...°F.....	141	Time immersed..... days..	22
Binder—		Water absorbed..... per cent..	14
Kind.....	W. G. P.	Average for first 4 days... do....	1.92
Laboratory No.....	4683	Specific gravity (apparent).....	1.114
Amount..... per cent..	8.5		
Weight of—			
Fuel briquetted... pounds..	88,000		
Briquets, average... do.....	0.485		

Chemical analyses of coal and briquets.

	Car sample.	Test 217.
Laboratory No.....	4590	4680
Proximate:		
Moisture..... per cent..	15.83	8.29
Volatile matter..... do.....	29.08	30.55
Fixed carbon..... do.....	39.20	47.01
Ash..... do.....	15.89	14.15
Sulphur..... do.....	2.54	2.38
Ultimate:		
Hydrogen..... do.....		4.43
Carbon..... do.....		68.76
Nitrogen..... do.....		1.19
Oxygen..... do.....		7.50

Extraction analyses.

	Pitch.	Fuel.	Briquets.
		Ind. No. 1B.	Test 217.
Laboratory No.	4683	4590	4680
Air-drying loss..... per cent.		13.90	3.60
Extracted by CS ₂ (as received)..... do.	89.31	.69	7.08
Pitch in briquet, by analysis..... do.			7.21
Heat value..... B. t. u.	16,637	9,866	11,444

INDIANA NO. 2.

A sample of bituminous run-of-mine coal from Boonville, Warrick County, on the Southern Railway, was designated Indiana No. 2 and a part of the sample was subjected to a briquetting test.

One ton of this coal was briquetted with 7 per cent of pitch H, the briquets being satisfactory in every way. They were brown in color and resembled those made of Indiana No. 1 coal, but were much stronger, by reason of the better quality of pitch. They weighed on the average 6.42 pounds each. No analysis of the briquets was made.

Chemical analysis.

	Car sample.		Car sample.
Laboratory No.	1495	Ultimate:	
Proximate:		Hydrogen..... per cent.	4.71
Air-drying loss..... per cent.	3.60	Carbon..... do.	67.16
Moisture..... do.	9.62	Nitrogen..... do.	1.33
Volatile matter..... do.	36.14	Oxygen..... do.	7.49
Fixed carbon..... do.	41.22	Heat value..... B. t. u.	11,122
Ash..... do.	13.02		
Sulphur..... do.	4.43		

INDIANA NO. 5B.

A sample of bituminous screenings (through 1½-inch bar screen) from Hymera, Sullivan County, on the Evansville & Terre Haute Railroad, was designated Indiana No. 5B and was used in briquetting tests 229 and 230.

The briquets made were identical in characteristics with those made from Indiana No. 1B and Indiana No. 6B coal.

Locomotive and house-boiler tests were made of the briquets from test 230, but locomotive steaming tests only were made of the briquets from test 229.

Briquetting tests.

	Test 229.	Test 230.
Size as shipped	Screenings.	Screenings.
Size as used:		
Over $\frac{1}{8}$ inch	1.0	1.0
$\frac{1}{8}$ inch to $\frac{1}{4}$ inch	5.6	5.6
$\frac{1}{4}$ inch to $\frac{3}{8}$ inch	11.8	11.8
$\frac{3}{8}$ inch to $\frac{1}{2}$ inch	20.4	20.4
Through $\frac{1}{2}$ inch	61.2	61.2
Details of manufacture:		
Machine used	English.	Ranfrow No. 1.
Briquetting temperature	176	140
Binder—		
Kind	W. G. P.	W. G. P.
Laboratory No.	4806	4806
Amount	7	9
Weight of—		
Fuel briquetted	24,000	30,000
Briquets, average	3.69	0.424
Drop test (1-inch screen):		
Heid	81.6	50.0
Passed	18.4	50.0
Tumbler test (1-inch screen):		
Heid	79.3	81.5
Passed	20.7	18.5
Fines through 10-mesh sieve	72.5	92.3
Absorption test:		
Time immersed	13	23
Water absorbed	7.4	13.8
Average for first 4 days	1.2	1.86
Specific gravity (apparent)	1.078	1.097

Chemical analyses of coal and briquets.

	Car sample.	Test 229.	Test 230.
Laboratory No.	4708	4822	
Proximate:			
Moisture	13.48	7.27	6.34
Volatile matter	31.94	37.73	37.63
Fixed carbon	42.97	44.37	44.74
Ash	11.61	10.63	11.39
Sulphur	4.08	4.29	4.76
Ultimate:			
Hydrogen	4.65	4.49	4.62
Carbon	67.30	68.47	68.06
Nitrogen	1.09	1.13	1.10
Oxygen	8.82	9.80	8.96

Extraction analyses.

	Pitch.	Fuel.	Briquets.	
		Ind. No. 5B.	Test 229.	Test 230.
Laboratory No.	4806	4708	4822	
Air-drying loss		10.00		
Extracted by CS ₂ (as received)	96.90		6.87	8.50
Pitch in briquet by analysis			5.82	7.52
Heat value	16,864	10,822	11,945	12,046

INDIANA NO. 6B.

Bituminous screenings, through $1\frac{1}{4}$ -inch bar screen, from vein No. 4 at Hymera, Sullivan County, on the Evansville & Terre Haute Railroad, were designated Indiana No. 6B, and were used in briquetting

tests 220 and 227. A locomotive steaming test was made of briquets from test 220 and a house-boiler steaming test of briquets from test 227.

Renfrow briquets were similar to those made from Indiana No. 1B coal. Seven per cent binder gave satisfactory English briquets when the coal was thoroughly dried; otherwise the briquets were soft and easily broken when cold. When warm, the briquets were soft and spongy, although cohesive enough to be handled without breaking. When cold, they became tough, with porous surfaces, rough fractures, and ragged edges.

Briquetting tests.

	Test 220.	Test 227.
Size as shipped.....	Screenings.	Screenings.
Size as used:		
Over $\frac{1}{2}$ inch..... per cent..	1.2	1.2
$\frac{1}{2}$ inch to $\frac{3}{4}$ inch..... do..	5.6	5.6
$\frac{3}{4}$ inch to $\frac{1}{2}$ inch..... do..	10.2	10.2
$\frac{1}{2}$ inch to $\frac{1}{4}$ inch..... do..	20.4	20.4
Through $\frac{1}{4}$ inch..... do..	62.6	62.6
Details of manufacture:		
Machine used.....	Renfrow No. 1.	English.
Briquetting temperature..... °F..	140	168
Binder—		
Kind.....	W. G. P.	W. G. P.
Laboratory No.....	4806	4806
Amount..... per cent..	8.5	7
Weight of—		
Fuel briquetted..... pounds..	64,000	8,000
Briquets, average..... do..	0.484	3.84
Drop test (1-inch screen):		
Held..... per cent..	58.5	83.7
Passed..... do..	41.5	16.3
Tumbler test (1-inch screen):		
Held..... do..	84.0	81.2
Passed..... do..	16.0	18.8
Fines through 10-mesh sieve..... do..	90.0	78.3
Absorption test:		
Time immersed..... days..	13	13
Water absorbed..... per cent..	12.5	6.7
Average for first 4 days..... do..	2.04	1.06
Specific gravity (apparent).....	1.063	1.156

Chemical analyses of coal and briquets.

	Raw fuel.	Test 220.	Test 227.
Laboratory No.....	4804	4835	4872
Proximate:			
Moisture..... per cent..	11.48	5.84	5.72
Volatile matter..... do..	35.63	38.08	37.83
Fixed carbon..... do..	40.63	44.22	43.16
Ash..... do..	12.26	11.86	13.29
Sulphur..... do..	5.08	4.49	4.87
Ultimate:			
Hydrogen..... do..	4.79	5.04	4.76
Carbon..... do..	66.96	70.61	66.48
Nitrogen..... do..	1.13	1.14	1.07
Oxygen..... do..	7.53	5.85	8.42

Extraction analyses.

	Pitch.	Fuel.	Briquets.	
		Ind. No. 6B.	Test 220.	Test 227.
Laboratory No.	4806	4804	4835	4872
Air-drying loss..... per cent.		7.00	3.30	2.70
Extract by CS ₂ (as received)..... do.	96.90	.76	8.10	7.47
Pitch in briquet by analysis..... do.			7.64	6.99
Heat value..... B. t. u.	16,864	10,967	12,132	11,839

INDIANA NO. 19.

Bituminous screenings, through 1½-inch stationary bar screen, from the Brazil Block top bed at Diamond, Parke County, on the Chicago & Eastern Illinois Railroad, were designated Indiana No. 19 and were used in briquetting test 165. Steaming test 464 was made of the briquets; a locomotive steaming test was also made of them.

The briquets made of the dry coal with 8 per cent of binder were hard, with firm surfaces and very rough fracture. With moist coal, satisfactory briquets were made regardless of the percentage of binder used. These briquets had a dull-gray color, due to the presence of a large quantity of clay.

Briquetting test.

	Test 165.		Test 165.
Size as shipped.....	Screenings.	Drop test (1-inch screen):	
Size as used:		Held..... per cent.	64.0
Over ½ inch..... per cent.	3.6	Passed..... do.	36.0
¼ inch to ½ inch..... do.	12.4	Tumbler test (1-inch screen):	
⅜ inch to ½ inch..... do.	20.0	Held..... do.	89.0
⅝ inch to ¾ inch..... do.	24.0	Passed..... do.	11.0
Through ¾ inch..... do.	40.0	Fines through 10-mesh sieve,	
Details of manufacture:		per cent.	91.5
Machine used.....	Renfrow No. 1.	Weather test:	
Briquetting temperature, ° F.	131	Time exposed..... days..	95
Binder--		Condition.....	C
Kind.....	W.G.P.	Absorption test:	
Laboratory No. 2..... {	3962	Time immersed..... days..	16
Amount..... per cent.	4120	Water absorbed..... per cent.	12.6
Weight of--		Average for first 4 days, per	
Fuel briquetted.. pounds..	80,000	cent.	1.90
Briquets, average... do.	0.562	Specific gravity (apparent).....	1.211

* Binder contained 6 parts No. 4120 and 1 part No. 3962.

Chemical analyses of coal and briquets.

	Raw fuel.	Test 165.
Laboratory No.	3979	4315
Proximate:		
Moisture..... per cent.	16.91	4.89
Volatile matter..... do.	26.85	30.53
Fixed carbon..... do.	38.87	44.00
Ash..... do.	17.37	20.58
Sulphur..... do.	1.89	3.36
Ultimate:		
Hydrogen..... do.	4.33	4.18
Carbon..... do.	63.75	61.71
Nitrogen..... do.	1.23	1.00
Oxygen..... do.	7.53	7.94

Extraction analyses.

	Pitches.		Fuel.	Briquets.
			Ind. No. 19.	Test 165.
Laboratory No.	3962	4120	3979	4315
Air-drying loss..... per cent.			13.10	5.40
Extract by CS_2 (as received)..... do.	77.79	97.70	1.12	7.39
Pitch in briquet by analysis..... do.				6.90
Heat value..... B. t. u.	16,946	16,946	9,524	10,649

INDIANA NO. 20.

Bituminous screenings, through a $1\frac{1}{4}$ -inch bar screen, from the Brazil Block bottom bed at Brazil, Clay County, on the Chicago & Eastern Illinois Railroad, were designated Indiana No. 20, and a part of the washed sample was used in briquetting test 169. A locomotive steaming test was made of the briquets.

The surfaces of briquets with 7 per cent of binder became soft and easily abraded as the briquets cooled. The briquets were easily transported from the press, but produced a high percentage of slack when handled afterwards. Their fracture was rough and their color brighter than in briquets from unwashed fuel.

Briquetting test.

	Test 169.		Test 169.
Size as shipped.....	Screenings.	Drop test (1-inch screen):	
Size as used:		Held..... per cent.	39.0
Over $\frac{1}{4}$ inch..... per cent.	0.0	Passed..... do.	61.0
$\frac{1}{4}$ inch to $\frac{1}{2}$ inch..... do.	3.0	Tumbler test (1-inch screen):	
$\frac{1}{4}$ inch to $\frac{3}{8}$ inch..... do.	16.2	Held..... per cent.	71.0
$\frac{3}{8}$ inch to $\frac{1}{2}$ inch..... do.	26.6	Passed..... do.	29.0
Through $\frac{1}{2}$ inch..... do.	54.2	Fines through 10-mesh sieve, per cent.	93.8
Details of manufacture:		Weather test:	
Machine used.....	Renfrow No. 1.	Time exposed..... days.	4 $\frac{1}{2}$
Briquetting temperature. ° F.	131	Condition.....	B
Binder—		Absorption test:	
Kind.....	W. G. P.	Time immersed..... days.	19
Laboratory No.....	4318, 4543	Water absorbed..... per cent.	28.5
Amount..... per cent.	7	Average for first 4 days. do.	4.0
Weight of—		Specific gravity (apparent).....	0.903
Fuel briquetted..... pounds.	30,000		
Briquets, average..... do.	0.427		

Chemical analyses of coal and briquets.

	Raw fuel.	Test 169.
Laboratory No.	3979	4627
Proximate:		
Moisture..... per cent.	16.91	9.67
Volatile matter..... do.	26.85	35.75
Fixed carbon..... do.	38.87	47.65
Ash..... do.	17.37	6.93
Sulphur..... do.	1.89	1.40
Ultimate:		
Hydrogen..... do.	4.33	5.06
Carbon..... do.	63.75	75.14
Nitrogen..... do.	1.22	1.25
Oxygen..... do.	7.53	9.33

Extraction analyses.

	Pitches.		Fuel.	Briquets.
			Ind. No. 20.	Test 160.
Laboratory No.	4318	4543	3979	4627
Air-drying loss.....per cent..			13.10	7.90
Extracted by C_2H_2 (as received).....do..	90.42	99.66	.63	8.40
Pitch in briquet, by analysis.....do..				8.39
Heat value.....B. t. u..	16,811	16,811	9,524	12,391

INDIAN TERRITORY NO. 2.

A sample of bituminous run-of-mine coal from Hartshorne, Pittsburg County, Okla. (Indian Territory on the date test was made), on the Rock Island system, was designated Indian Territory No. 2, and a part of the sample was subjected to briquetting tests.

This coal was tested with the hard pitch X. With 8.5 per cent of pitch the briquets were not strong and crumbled readily. With 12 per cent of pitch the briquets were not much better, and with 13.4 per cent they looked well as they came from the press but were not strong and began to lose their corners soon after being exposed to the weather. Judging from the runs made, this coal will require more than the usual pressure. The briquets containing 8.5 per cent of pitch weighed only 5.08 pounds each. Those with 13.4 per cent of pitch weighed 6.15 pounds each.

Chemical analysis.

	Car sample.		Car sample.
Laboratory No.	1184	Ultimate:	
Proximate:		Hydrogen.....do..	4.90
Air-drying loss.....per cent..	2.80	Carbon.....do..	72.73
Moisture.....do..	4.45	Nitrogen.....do..	1.75
Volatile matter.....do..	36.15	Oxygen.....do..	7.52
Fixed carbon.....do..	48.40	Heat value.....B. t. u..	12,607
Ash.....do..	11.00		
Sulphur.....do..	1.62		

INDIAN TERRITORY NOS. 2B AND 2C.

Bituminous slack screenings, through a $\frac{1}{8}$ -inch shaking screen, from the Hartshorne bed at Hartshorne, on the Rock Island Railroad, were designated Indian Territory No. 2B, and a part of this sample was used in briquetting tests 135, 136, 137, 138, 139, 145, 146, 147, 148, 149, 157, and 168. Steaming tests 453 and 456 were made of the briquets from briquetting tests 137, 148, and 157. Locomotive steaming tests were also made of briquets from briquetting tests 148 and 149. House-boiler steaming tests were made of briquets from tests 148 and 157.

Bituminous lump coal, over 1-inch shaking screen, from the same locality as Indian Territory No. 2B coal, was designated Indian Territory No. 2C, and a part of the sample was used in briquetting tests 153, 154, 155, and 156. No steaming tests or locomotive or house-boiler tests were made of briquets from this sample.

In tests 135, 136, 137, 145, 146, 147, 148, and 168, with Indian Territory No. 2B raw slack coal, there was no apparent difference in English briquets with 6, 7, and 8 per cent of binder. A small lot with 5 per cent of binder showed shortage of pitch. The pressure on the machine was changed during the making of Renfrow briquets, as indicated in the results of the tests. All briquets were satisfactory, having sharp edges, hard surfaces, and clean, rough fracture; they could be handled while warm from the machine and could be piled 5 feet high without sticking together or crushing.

Briquetting tests of Indian Territory No. 2B coal.^a

	Test No.							
	135.	136.	137.	145.	146.	147.	148.	168.
Details of manufacture:								
Machine used....	English.	English.	English.	English.	English.	English.	Renfrow No. 1.	Renfrow No. 1.
Briquetting temperature (*F.)..	185	185	185	185	185	185	149	149
Binder—								
Kind.....	W. G. P.	W. G. P.	W. G. P.	W. G. P.	W. G. P.	W. G. P.	W. G. P.	W. G. P.
Laboratory No.....	3258	3258	3296	3486	3623	3624	3486	3296
Amount, per cent....	6	7	8	6	7	8	8	8
Weight of—								
Fuel briquetted, lbs.	11,000	22,000	70,000	16,000	16,000	16,000	19,400	99,000
Briquets, average, lbs..	3.46	3.88	3.77	3.53	3.64	3.43	0.515	0.506
Drop test (1-inch screen):								
Held... per cent....				81.7	82.8	84.2	58.5	
Passed... do.....				18.3	17.2	15.8	41.5	
Tumbler test (1-inch screen):								
Held... per cent....		74.8		73.3	79.8	80.8	81.0	
Passed... do.....		25.2		26.7	20.2	19.2	19.0	
Fines through 10-mesh sieve, per cent.....		75.0		86.5	85.6	75.5	95.4	
Weathering test:								
Time exposed, days.....	239	239	239	145	145	145	144	239
Condition.....	B	C	B	B	A	A	B	B
Absorption test:								
Time immersed, days.....				16	16	16	16	
Water absorbed, per cent.....				8.6	10.2	11.4	12.9	
Average for first 13 days, per cent.....				1.62	.72	.84		
Average for first 6 days, per cent.....							1.7	
Specific gravity (apparent).....				1.151	1.133	1.076	1.117	

^a Raw slack.

As stated above, tests 138, 139, 149, and 157 were made with Indian Territory No. 2B washed slack coal, and tests 153, 154, 155, and 156 with Indian Territory No. 2C lump coal. Tests 138 and 139 were made with the first sample sent to the plant and tests 149 and 157 with the second sample. Excellent briquets were made with 8 per cent of binder. The English briquets showed the advantage of higher pressure by being harder and closer grained and having smoother surfaces and sharper edges. Similar differences marked the superiority of these briquets over those made from the same slack unwashed.

The briquets made from the lump coal (Indian Territory No. 2B) may be considered as standing in value between those made from washed and those made from unwashed slack from the same mine; they are identical in appearance.

Briquetting tests with Indian Territory No. 2B^a and Indian Territory No. 2C^b coal.

	Test No.							
	138.	139.	149.	157.	153.	154.	155.	156.
Details of manufacture:								
Machine used.....	English.	Renfrow No. 1.	Renfrow No. 1.	English.	English.	English.	English.	Renfrow No. 1.
Briquetting temperature (°F.)...	185	149	149	185	185	185	185	149
Binder—								
Kind.....	W. G. P.	W. G. P.	W. G. P.	W. G. P.	W. G. P.	W. G. P.	W. G. P.	W. G. P.
Laboratory No.....	3624	3296	3624	3624	3486	3486	3486	3486
Amount, per cent....	8	8	8	8	6	7	8	8
Weight of—								
Fuel briquetted, lbs	16,000	20,000	40,000	8,000	1,000	1,000	1,000	8,000
Briquets, average, lbs..	3.31	0.451	0.476	3.34	3.11	3.19	3.14	0.490
Drop test (1-inch screen):								
Held... per cent...	88.9	60.0	56.0	85.5	79.8	84.6	86.4	70.0
Passed...do....	11.1	40.0	44.0	14.5	20.2	15.4	13.6	30.0
Tumbler test (1-inch screen):								
Held... per cent...	85.0	79.0	76.0	82.5	71.0	76.6	83.5	81.5
Passed (fines), per cent.....	15.0	21.0	24.0	17.5	29.0	23.4	16.5	18.5
Fines through 10-mesh sieve, per cent.....	79.5	90.5	93.7	78.1	82.3	80.6	78.4	90.9
Weathering test:								
Time exposed, days.....	128	127	149	128	239	239	239	182
Condition.....	B	C	B	A	C	C	C	B
Absorption test:								
Time immersed, days.....	19	19	19	19	19	19	15	19
Water absorbed, per cent.....	9.6	14.7	16.0	10.3	12.6	11.1	10.2	13.5
Average for first 5 days, per cent...	1.20	2.0	2.26	1.06		1.34	1.57	1.84
Average for first 12 days, per cent.....					0.93			
Specific gravity (apparent).....	1.097	1.061	1.024	1.086	1.070	1.084	1.088	1.077

^a Washed slack.

^b Lump.

Chemical analyses of raw fuel and briquets.

	Raw fuel.	Raw fuel.	Tests 135 and 136.	Test 137.	Tests 138 and 157.
Laboratory No.....	3381	3472	3362	3342	4108
Proximate:					
Moisture.....per cent..	3.77	5.09	3.61	3.85	2.85
Volatile matter.....do...	32.65	32.45	33.03	34.05	34.91
Fixed carbon.....do....	51.15	48.76	49.64	49.26	54.03
Ash.....do.....	12.43	13.10	13.72	12.73	8.21
Sulphur.....do.....	1.79	1.07	1.76	1.71	1.50
Ultimate:					
Hydrogen.....do.....	4.77	4.53	4.65	4.54	4.87
Carbon.....do.....	72.37	72.05	71.08	73.17	76.28
Nitrogen.....do.....	1.77	1.80	1.46	1.61	1.71
Oxygen.....do.....	6.31	5.96	6.75	5.66	7.15

	Test 139.	Tests 145, 146, and 147.	Test 148.	Test 149.	Test 168.
Laboratory No.....	3896	3520	4086	3550	3343
Proximate:					
Moisture.....per cent..	2.80	2.87	3.29	2.13	3.85
Volatile matter.....do...	35.67	35.02	35.25	36.97	34.05
Fixed carbon.....do....	53.77	50.60	53.15	54.14	49.36
Ash.....do.....	7.76	11.51	8.31	6.76	12.73
Sulphur.....do.....	1.58	1.72	1.52	1.47	1.71
Ultimate:					
Hydrogen.....do.....	4.91	4.48	4.93	5.04	4.54
Carbon.....do.....	75.63	73.39	76.11	77.98	73.17
Nitrogen.....do.....	1.52	1.49	1.69	1.78	1.61
Oxygen.....do.....	8.33	7.02	7.11	6.79	5.66

Extraction analyses.

	Fuels.		Briquets.			
	Ind. T. No. 2B.	Ind. T. No. 2C.	Tests 135 and 136.	Test 137.	Tests 138 and 157.	Test 139. Tests 145, 146, and 147.
Laboratory No.....	3381	3472	3262	3342	4108	3896
Air-drying loss.....per cent..	1.50	3.80	1.50		0.90	1.80
Extracted by CS_2 (as received), per cent.....	.32	.41	4.72	6.32	6.04	7.58
Pitch in briquet by analysis, per cent.....			5.50	7.32	6.32	8.75
Heat value.....B. t. u..	12,400	12,033	12,317	12,481	13,562	13,541

	Pitches.		Briquets.		
			Test 148.	Test 149.	Test 168.
Laboratory No.....	3624	3298	3486	3258	4086
Air-drying loss.....per cent..					1.30
Extracted by CS_2 (as received), per cent.....	89.60	82.43	85.57	80.27	6.00
Pitch in briquet by analysis, per cent.....					6.57
Heat value.....B. t. u..	16,193	16,427	16,407	16,373	13,412

INDIAN TERRITORY NO. 3.

A sample of bituminous run-of-mine coal from Edwards, Ind. T. (now Oklahoma), on the Rock Island Railroad, was designated Indian Territory No. 3, and a part of the sample was subjected to briquetting tests at the St. Louis plant.

This coal, which is hard and glossy, was tested on the English machine with 8 per cent of pitch A. The mixture made fairly good briquets, which were clean and hard, although somewhat porous, but they were not so strong as was desired. This defect was due in part to lack of sufficient heat to soften the pitch thoroughly in the pug mill. The briquets as received from the machine weighed 6.56 pounds each. With 7.2 per cent of pitch A, this coal made smooth and fairly hard briquets, which weighed 6.71 pounds each. With 6 per cent of pitch A, briquets were made that were smooth and well pressed, but soft. These averaged in weight 6.8 pounds each. If the pitch could have been heated in the English machine to a temperature sufficiently high to cause it to soften and mix with the coal the mixture would undoubtedly have made a good briquet, because with a soft pitch, as pitch D or pitch H, this coal could be briquetted with 6 per cent of binder.

Chemical analysis.

	Car sample.		Car sample.
Laboratory No.	1274	Ultimate:	
Proximate:		Hydrogen.....per cent..	4.62
Air-drying loss.....per cent..	1.20	Carbon.....do....	70.62
Moisture.....do....	4.61	Nitrogen.....do....	1.55
Volatile matter.....do....	37.00	Oxygen.....do....	7.72
Fixed carbon.....do....	47.25	Heat value.....B. t. u..	12,319
Ash.....do....	11.14		
Sulphur.....do....	3.63		

INDIAN TERRITORY NO. 6.

A sample of bituminous slack coal from Coalgate, Ind. T. (now Oklahoma), on the Missouri, Kansas & Texas Railway, was designated Indian Territory No. 6, and a part of the sample was subjected to briquetting tests at the St. Louis plant.

The sample consisted of a carload of dirty slack which contained considerable fire clay and some shale. Laboratory experiments had shown that 7 per cent of a binder of the quality of pitch D was the minimum that could be expected to give satisfactory results. Accordingly, 1 ton of this coal was mixed by hand with 7 per cent of pitch D, but the briquets were crumbly and not at all satisfactory, although dense and well pressed. They weighed 7.35 pounds each. They were full of gray streaks on the surface, due to the fire clay, and on standing became coated with a heavy gray efflorescent substance which was found to be calcium sulphate. Eggettes of this same mixture made on the American machine were practically the same in character and unusually dull gray in appearance. The specific gravity of the briquets was 1.205. When tested on the compression machine these briquets, which were crumbly and weak and broke

easily in the hands, compressed much more than usual before breaking. Their crushing strength was 5,600 pounds, or 195 pounds per square inch. Another ton was tried with 9 per cent of pitch D, but the briquets were practically the same as with 7 per cent, although a little darker in color. They also showed similar streaks of fire clay on the surface and their fracture was decidedly earthy. They weighed on the average 7.5 pounds each. There seems to have been little adhesiveness between this pitch and the dirty coal, and to make good briquets with this pitch it would probably be necessary to use as much as 15 per cent, if not more.

Satisfactory results with soft pitch not having been obtained, 1 ton of the slack coal was tried with 8 per cent of the harder pitch A. The briquets were well pressed and blacker than those with 7 per cent of pitch D, but were not so strong. The mixture did not show any tendency to stiffen in the machine. All of the tests with this coal demonstrated its injurious effect on the binding properties of the pitch.

The coal was next tried by mixing together 93 per cent of coal, 6 per cent of rosin, and 1 per cent of quicklime. The lime was mixed in sizes ranging from dust to pieces three-fourths of an inch in diameter, and it was thought that the lumps would be reduced by the disintegrator and the lime slaked by the steam of the pug mill, so that the cement would not set too quickly. The disintegrator, however, had little effect on the hard lime, and consequently there were in the briquets many large unslaked pieces, which caused the briquets to fall to pieces. Some of the material that remained in the pug mill for some time gave a briquet that was hard while hot, but after cooling became earthy. The briquets weighed on an average 7 pounds each. The eggettes made of this material were dense but crumbly.

In order to prevent unslaked lime from going into the briquets, the lime before being used was slaked with one-half its weight of water, making a fine, dry powder. This was mixed with the coal to give the same proportions with the rosin as in the above test, and although the briquets were smooth and well formed, having been thoroughly pressed, they were crumbly and not at all strong. They were grayish in color and similar in quality to those made with 7 per cent of pitch D. They weighed on an average 7.5 pounds each. The eggettes of this same material were also crumbly and dull in luster.

Some of the lime-rosin briquets were burned and gave less smoke than those with the rosin and no lime, some giving even less than the briquets made with pitch alone.

Another test of this coal was made with rosin and Kansas crude oil. The experiments in the laboratory had shown that the best results could be obtained by using 6 per cent of rosin and 1 per cent

of the oil. In preparing this combination for the English machine the oil was first mixed with five times its weight of fine coal, so as to make a nonsticky mass that would readily mix with the cracked rosin and coal in the usual manner. As the rosin became very fluid at 212° F. the hot briquets when received from the machine had no strength, and cracked slightly under their own weight when piled only two high. They also cracked somewhat because of the water in the wet steam. On cooling they became strong and were of good quality. Some of the mixture that was allowed to stand in the mold until cool enough to be "tacky" before pressing was good as received from the machine, but it had not been sufficiently compacted. On standing, these briquets became coated with a heavy brownish-gray efflorescence. The weight of these briquets averaged 7.29 pounds each and their specific gravity was 1.25. The crushing strength of the briquets was on the average 6,400 pounds, or 223 pounds per square inch. On account of the lack of uniformity of the briquets, however, the results of eight tests varied from 4,300 to 9,240 pounds. The eggettes made from this same mixture were hard and compact, although of a dull color, and they had a good degree of strength as received from the machine. With an arrangement for cooling the mass after it had passed through the pug mill and before being fed to the press this mixture made briquets that were very good physically, but smoked rather badly.

As the above tests showed that the best way to counteract the effect of the large amount of fire clay in this coal was to add some liquid binder that would soak into the coal and at the same time unite with the solid binder, another test was made with creosote as a liquid binder and pitch as the hard binder. For this purpose Indian Territory No. 6 coal was mixed with 8 per cent of pitch G and 1½ per cent of creosote such as is used in preserving timber. The creosote and five times its weight of fine coal were first mixed together and then thoroughly mixed with the cracked pitch and coal. In operating the machine the amount of steam used for heating the mixture was gradually increased until there was an excess. This, however, had no effect upon the cooled briquets, but it made the hotter ones more tender and a little harder to handle. The first briquets of this mixture cracked a little, because of excess of water, but the remainder came from the machine strong enough to bear handling at once. The effect of the excess of steam was to insure more complete softening of the pitch and more uniform mixing of the clay. The briquets were black and clean and only slightly streaked with the clay. When piled they stuck together slightly, and were mottled a little from the creosote. However, they showed much less of the gray efflorescent substance than any of the Indian Territory No. 6 briquets and were also the best made, although

earthy. They were smooth and black, and although crumbling little or not at all in handling, they would crumble a great deal on breaking or firing. On the other hand, the eggettes, made of this same mixture on the American machine, were just as strong and would need no breaking in firing. The weight of the briquets was 7.25 pounds each on an average. Their specific gravity was 1.215 and their crushing strength was 6,500 pounds, or 226 pounds to the square inch.

Another test with the Indian Territory No. 6 coal was made with asphalt B5, which during the cool weather had become hard enough to crack, and thus could be readily used on the English machine. One ton of this coal with 8 per cent of asphalt made soft briquets that were hard to handle while warm, especially when they contained moisture due to the wet steam. The better ones were compact, heavy, and tough, and although they had an earthy, oily appearance, did not crack when struck a severe blow. They would stand rough handling, but would be improved with a small percentage of rosin. The briquets weighed 7.12 pounds each, and had a specific gravity of 1.22. Their crushing strength was very low, averaging 3,200 pounds, or 111 pounds per square inch. It varied from 1,800 to 5,100 pounds. In making the compression tests the briquets yielded a great deal before breaking.

A sample of pitch H was tried with the Indian Territory No. 6 coal. The best results were obtained with this as a binder. One ton of coal was mixed with 8 per cent of pitch H, and with nearly dry steam and good pressure the mixture made hard, compact briquets. Although somewhat earthy, they would stand rough handling. Their average weight was 7.27 pounds each.

Proximate analysis.

	Car sample.		Car sample.
Laboratory No.....	1596	Fixed carbon..... per cent..	41.40
Air-drying loss..... per cent..	3.50	Ash..... do.....	19.29
Moisture..... do.....	8.03	Sulphur..... do.....	3.20
Volatile matter..... do.....	31.28		

INDIAN TERRITORY NO. 9.

Semibituminous run-of-mine coal from Panama, on the Kansas City Southern Railroad, was designated Indian Territory No. 9, and a part of the sample was used in briquetting test 167. Steaming test 450 was made of the briquetted fuel.

Excellent briquets were made with 7 per cent of pitch, a quantity that might be reduced if more pressure were used. The briquets

were hard, with smooth surfaces and glossy, clean fracture; they were easily handled while hot from the machine, but became somewhat brittle when cold.

Briquetting test.

	Test 167.		Test 167.
Size as used:		Drop test (1-inch screen):	
Over $\frac{1}{8}$ inch..... per cent..	0.8	Held..... per cent..	62.5
$\frac{1}{8}$ inch to $\frac{1}{4}$ inch..... do....	3.6	Passed..... do.....	37.5
$\frac{1}{4}$ inch to $\frac{3}{8}$ inch..... do....	12.6	Tumbler test (1-inch screen):	
$\frac{3}{8}$ inch to $\frac{1}{2}$ inch..... do....	25.2	Held..... do.....	82.5
Through $\frac{1}{2}$ inch..... do....	57.8	Passed..... do.....	17.5
Details of manufacture:		Fines through 10-mesh sieve,	
Machine used.....	Renfrow No. 1.	per cent.....	98.0
Briquetting temperature. °F..	176	Weather test:	
Binder—		Time exposed..... days..	31
Kind.....	W. G. P.	Condition.....	B
Laboratory No.....	4120	Absorption test:	
Amount..... per cent..	7	Time immersed..... days..	18
Weight of—		Water absorbed..... per cent..	14.6
Fuel briquetted.. pounds..	70,000	Average for first 4 days..... do....	2.62
Briquets, average.. do....	0.459	Specific gravity (apparent).....	1.090

Chemical analyses of coal and briquets.

	Our sample.	Test 167.
Laboratory No.....	4020	4325
Proximate:		
Moisture..... per cent..	5.11	3.39
Volatile matter..... do....	13.65	17.15
Fixed carbon..... do....	73.21	72.04
Ash..... do....	8.03	7.42
Sulphur..... do....	1.18	1.15
Ultimate:		
Hydrogen..... do....	4.30	4.23
Carbon..... do....	82.59	82.89
Nitrogen..... do....	1.69	1.61
Oxygen..... do....	1.72	2.40

Extraction analyses.

	Pitch.	Fuel.	Briquets.
		Ind. T. No. 9.	Test 167.
Laboratory No.....	4120	4020	4325
Air-drying loss..... per cent..		4.50	0.90
Extracted by C_6H_6 (as received)..... do....	97.70	.48	6.14
Pitch in briquet by analysis..... do....			5.83
Heat value..... B. t. u..	17,060	13,662	14,103

IOWA NO. 4.

A sample of bituminous coal, over $1\frac{1}{8}$ -inch screen, from Center-ville, Appanoose County, on the Chicago, Burlington & Quincy Railroad, and four other railroads, was designated Iowa No. 4, and a portion of the sample was subjected to briquetting tests at the St. Louis plant.

One ton of this coal was briquetted with 7 per cent of pitch E. The briquets were well pressed and of a grayish color, but on cooling crumbled decidedly. They weighed 6.73 pounds each. As they did

not contain an excess of pitch, 7 tons more of this coal was briquetted with 8 per cent of pitch E, in order to have a sufficient quantity for a steaming test. The resultant briquets were bluish black in color, but were not quite hard enough, although fairly strong; they would stand considerable rough treatment in transportation. In burning they held together until consumed. They weighed on an average 6.77 pounds each. The eggettes made from this same mixture were stronger than the briquets, had a polished surface, but were very brown in color. In the cookstove they burned satisfactorily, crumbling little until consumed. The steaming test made of these briquets gave considerably better results than the original coal, indicating that the coal had been much improved by briquetting.

Chemical analyses of coal and briquets.

	Car sample.	Briquets.		Car sample.	Briquets.
Laboratory No.....	1437	1488	Ultimate:		
Proximate:			Hydrogen.....per cent..	4.67	4.81
Air-drying loss.....per cent..	4.50	3.90	Carbon.....do.....	68.07	69.25
Moisture.....do.....	14.08	13.24	Nitrogen.....do.....	1.05	.96
Volatile matter.....do.....	35.59	36.50	Oxygen.....do.....	8.49	6.28
Fixed carbon.....do.....	39.37	37.85	Heat value.....B. t. u..	10,723	10,885
Ash.....do.....	10.96	12.41			
Sulphur.....do.....	4.26	3.90			

KANSAS NO. 2.

A sample of bituminous lump and nut coal from Yale, Crawford County, on the Missouri Pacific Railroad, was designated Kansas No. 2, and a part of the sample was subjected to briquetting tests at the St. Louis plant.

Two tons of this coal was briquetted with 11 per cent of pitch X. The pitch of the mixture, in passing through the machine from the pug mill to the press, set so quickly that the briquets did not receive the necessary pressure. The surface was rough, and a great many cracks developed. No more of this coal was available for briquetting after the receipt of pitch of the better qualities. It is, however, a coal that will briquet very readily with pitch of the better grades, such as pitch D or pitch H. About 7 per cent of either would be required for commercial briquets.

Chemical analysis.

	Car sample.		Car sample.
Laboratory No.....	1122	Ultimate:	
Proximate:		Hydrogen.....per cent..	4.41
Air-drying loss.....per cent..	2.00	Carbon.....do.....	64.58
Moisture.....do.....	4.18	Nitrogen.....do.....	.96
Volatile matter.....do.....	31.23	Oxygen.....do.....	4.82
Fixed carbon.....do.....	46.68	Heat value.....B. t. u..	11,642
Ash.....do.....	17.91		
Sulphur.....do.....	6.27		

KANSAS NO. 2B.

Bituminous slack coal, from the lower Weir-Pittsburg bed at Yale, Crawford County, was designated Kansas No. 2B and was used in briquetting tests 182, 183, 194, 195, 199, 203, and 204. The briquets were used in steaming tests 487, 488, and 495, and in locomotive and house-boiler steaming tests.

In tests 182, 183, and 199 raw slack was used as follows: Over one-fourth inch, 1 per cent; one-tenth inch to one-fourth inch, 4.8 per cent; one-twentieth inch to one-tenth inch, 12.6 per cent; one-fortieth inch to one-twentieth inch, 23.6 per cent; through one-fortieth inch, 58 per cent. The briquets showed the same characteristics as those made from washed coal, except that they were harder when cold, and had a harder and rougher fracture, owing to high ash content.

In tests 194, 195, 203, and 204 washed slack was used as follows: Over one-fourth inch, 0.8 per cent; one-tenth inch to one-fourth inch, 4.8 per cent; one-twentieth inch to one-tenth inch, 12.6 per cent; one-fortieth inch to one-twentieth inch, 21.4 per cent; through one-fortieth inch, 60.4 per cent. Satisfactory English briquets were made with 7 per cent of binder, and satisfactory Renfrow briquets with 8 and 9 per cent of binder, there being no apparent difference in these two kinds. Briquets with either had smooth outer surfaces, were well molded, with sharp edges, clean, rough fracture, and did not break in handling while warm. After the briquets became cold those with 9 per cent of binder showed less effect of abrasion than those with 8 per cent of binder. Renfrow briquets showed deficiency of pressure.

Briquetting tests.

	From raw slack.			From washed slack.			
	Test 182.	Test 183.	Test 199.	Test 194.	Test 195.	Test 203.	Test 204.
Details of manufacture:							
Machine used.....	(a)	(b)	(b)	(a)	(a)	(b)	(b)
Temperature of briquets, ° F.	158	149	149	176	176	149	149
Binder—							
Kind.....	W. G. P.	W. G. P.	W. G. P.	W. G. P.	W. G. P.	W. G. P.	W. G. P.
Laboratory No.....	4543	4543	4683	4543	4683	4683	4683
Amount, per cent.....	7	7	8	7	7	8	9
Weight of—							
Fuel briquetted, pounds	40,000	40,000	68,000	12,000	24,000	16,000	24,000
Briquets, average, pounds.....	3.95	0.482	0.455	3.65	3.61	0.441	0.417
Drop test (1-inch screen):							
Held, per cent.....	89.1	69.0	61.0	84.9	81.4	64.0	52.5
Passed, per cent.....	10.9	31.0	39.0	15.1	18.6	36.0	47.5
Tumbler test (1-inch screen):							
Held, per cent.....	85.2	91.0	88.5	80.8	83.5	84.5	80.5
Passed (fines), per cent.....	14.8	9.0	11.5	19.2	16.5	15.5	19.5
Fines through 10-mesh sieve, per cent.....	71.6	94.7	97.5	87.2	74.6	94.7	94.3
Weathering test:							
Time exposed, days.....	12	12		4	3		
Condition.....	B	A		A	A		
Absorption test:							
Time immersed, days.....	19	19	13	13	13	13	13
Water absorbed, per cent.....	9.2	12.6	12.4	8.7	9.7	13.6	12.9
Average for first 4 days, per cent.....	1.15	1.95	2.23	1.35	1.43	2.50	2.23
Specific gravity (apparent).....	1.198	1.170	1.143	1.122	1.105	1.044	1.098

a English.

b Renfrow No. 1.

Chemical analyses of coal and briquets.

	Car sample.	Raw fuel.	Test 182.	Test 183.	Test 199.
Laboratory No.....	4361	4518	4374	4380	4065
Proximate:					
Moisture.....per cent.	8.01	9.53	9.43	4.64	2.78
Volatile matter.....do.	28.39		29.71	28.40	31.67
Fixed carbon.....do.	45.22		43.67	45.56	46.78
Ash.....do.	20.38	10.87	17.19	30.40	18.77
Sulphur.....do.	4.70	3.80	4.15	4.49	4.36
Ultimate:					
Hydrogen.....do.	4.15		4.28	5.03	4.35
Carbon.....do.	83.62		86.12	84.05	86.00
Nitrogen.....do.	1.13		1.03	1.20	.96
Oxygen.....do.	3.83		5.01	3.62	4.80
		Test 194.	Test 195.	Test 203.	Test 204.
Laboratory No.....	4422	4060	4654	4655	
Proximate:					
Moisture.....per cent.	7.64	4.23	2.89	3.20	
Volatile matter.....do.	32.40	32.64	33.04	33.96	
Fixed carbon.....do.	51.52	53.67	52.26	51.53	
Ash.....do.	8.35	9.66	11.81	11.32	
Sulphur.....do.	3.60	3.36	3.84	3.82	
Ultimate:					
Hydrogen.....do.	4.61	4.81	4.32	4.55	
Carbon.....do.	74.58	76.20	72.18	73.26	
Nitrogen.....do.	1.25	1.20	1.09	1.12	
Oxygen.....do.	6.62	4.34	6.33	5.40	

Extraction analyses.

	Fuel.		Briquets.		
	Car sample.	Raw fuel.	Test 182.	Test 183.	Test 199.
Laboratory No.....	4361	4518	4374	4380	4065
Air-drying loss.....per cent..	6.60	8.20	7.00	3.00	-----
Extracted by CS ₂ (as received).....do.....	0.47	0.54	6.62	6.78	6.61
Pitch in briquet by analysis.....do.....	-----	-----	6.21	6.35	6.91
Heat value.....B. t. u.....	10,640	13,243	11,043	11,221	11,795

	Pitches.		Briquets.			
			Test 194.	Test 195.	Test 203.	Test 204.
Laboratory No.....	4543	4683	4422	4660	4654	4655
Air-drying loss.....per cent.....	-----	-----	6.20	-----	-----	-----
Extracted by CS ₂ (as received).....do.....	99.66	89.31	6.70	6.60	7.02	7.83
Pitch in briquet by analysis.....do.....	-----	-----	6.30	6.83	7.30	8.21
Heat value.....B. t. u.....	16,969	16,637	12,841	12,982	13,012	13,104

KENTUCKY NO. 1.

A sample of bituminous run-of-mine coal from Straight Creek, Bell County, on the Louisville & Nashville Railroad, was designated Kentucky No. 1, and a part of the sample was subjected to briquetting tests at the St. Louis plant.

This is a good coking coal, unusually light in weight and lustrous. In passing through the disintegrator it was ground a little too fine, even when only one side of the machine was used. Furthermore, on account of its light weight it was difficult to obtain the right pressure on the English machine. The coal was tested with 6 per

cent of pitch D, the briquets being clean and black, but not sufficiently pressed to fill all the pores. They were, however, good briquets, and would stand rough treatment in transportation. They stood weathering with little change. They weighed on an average 6.15 pounds each. The eggettes made from the mixture were smooth and lustrous and as strong as any eggettes made on the American machine. In the cookstove they burned satisfactorily without disintegrating or giving off an excess of smoke. This coal can be briquetted readily.

Chemical analysis.

	Car sample.		Car sample.
Laboratory No.....	1474	Ultimate:	
Proximate:		Hydrogen..... per cent..	5.25
Air-drying loss..... per cent..	1.30	Carbon..... do....	79.85
Moisture..... do....	3.10	Nitrogen..... do....	1.89
Volatile matter..... do....	36.12	Oxygen..... do....	7.22
Fixed carbon..... do....	56.39	Heat value..... B. t. u..	14,148
Ash..... do....	4.39		
Sulphur..... do....	1.22		

KENTUCKY NO. 2.

A sample of bituminous coal, over $\frac{1}{4}$ -inch screen, from Earlington, Hopkins County, on the Louisville & Nashville Railroad, was designated Kentucky No. 2, and a part of the sample was subjected to briquetting tests on both the English and the American machines at the St. Louis plant.

As none of the good pitches was available at the time this coal was tested, it was briquetted on the English machine with 9 per cent of pitch E, the necessary percentage as determined by laboratory work. Three tons of this coal was tested. The briquets were extremely pitchy, and, on account of excessive water due to wet steam, were difficult to handle, and when piled three or four deep crushed under their own weight. With dry steam the briquets were more easily handled. Although somewhat plastic, they did not crush when piled. On cooling, the briquets were hard, with lustrous fracture, and were capable of standing a great deal of rough handling. They weighed on an average 6.83 pounds each. The eggettes made of the mixture were hard, but almost too brittle. In the cookstove the eggettes burned satisfactorily.

The coal was tested with 8 per cent of pitch E, $3\frac{1}{4}$ tons of coal being used and the briquets being given the greatest pressure possible on the English machine. As the coal is hard, and as only one side of the disintegrator was used, there were many large pieces of coal in the briquets, but there were also sufficient small pieces to fill in all the openings between the larger fragments. This is nearly an ideal condition for briquetting on the English machine. The briquets were bluish black and excellent in every way. They did not have

the glossy or lustrous fracture of the briquets containing 9 per cent of pitch, but they were a better commercial briquet and were more easily handled when taken from the machine. With pitch of the quality of pitch D or pitch H, 5 to 6 per cent would make a good briquet. The briquets tested would stand much rough treatment in transportation. They did not disintegrate in burning, nor was there much slack formed in breaking them for burning. They weighed on an average 6.83 pounds each and had a specific gravity of 1.13, as compared with 1.37 for the slack or fine coal. The slack that was used in making the briquets contained 18 per cent of ash, but the lump coal contained only 11 per cent of ash. The crushing strength of the briquets was 11,300 pounds, or 394 pounds per square inch.

The results of the steaming test of the briquets showed that the briquets have no greater efficiency than the original coal.

The eggettes made from this same mixture were stronger and better than those made with 9 per cent of pitch, although not so hard. They were also browner in color. The eggettes were not so strong as the briquets.

Chemical analyses of coal and briquets.

	Car sample.	Briquets.
Laboratory No.....	1461	1544
Proximate:		
Air-drying loss..... per cent..	2.70	3.30
Moisture..... do.....	7.91	7.11
Volatile matter..... do.....	37.94	37.07
Fixed carbon..... do.....	45.03	44.31
Ash..... do.....	9.13	11.51
Sulphur..... do.....	3.62	2.71
Ultimate:		
Hydrogen..... do.....	5.00	4.50
Carbon..... do.....	71.46	69.99
Nitrogen..... do.....	1.32	1.28
Oxygen..... do.....	8.28	7.64
Heat value..... B. t. u....	12,200	11,878

KENTUCKY NO. 2B.

Coke breeze from Earlington, Hopkins County, on the Louisville & Nashville Railroad, was designated Kentucky No. 2B, and one part of the sample was used in briquetting test 102, and another part mixed with Illinois No. 20 coal was used in briquetting tests 103 and 104. No chemical analyses were made for briquetting test 102. Briquetting tests 103 and 104 are included under the discussion of Illinois No. 20.

In test 102 the size of the coal as used was as follows: Over one-fourth inch, 10.25 per cent; one-tenth inch to one-fourth inch, 31 per cent; one-twentieth inch to one-tenth inch, 18.75 per cent; one-fortieth inch to one-twentieth inch, 18.55 per cent; through one-fortieth inch, 21.45 per cent. The English machine was used, the

temperature of the briquets being 179.6° F. A coal-tar pitch and a water gas pitch binder were used, the proportions being 6 and 7 per cent. Twenty-five tons of fuel was briquetted.

The briquets were soft when hot, but on cooling were hard enough to stand handling. Various mixtures made no apparent difference. All briquets showed coarse structure and were easily fractured. Attempted boiler tests were discontinued, owing to difficulty in maintaining steam. Broken briquets burned better than whole ones. For pitch analyses, see laboratory Nos. 2729, 2735, and 2748. All of these briquets were sent to the Big Four Railway Co. for preliminary locomotive test. The results were unsatisfactory.

MARYLAND NO. 2.

Bituminous run-of-mine coal from the Big Vein, or Pittsburgh bed, one-half mile west of Lord, Allegany County, on the Baltimore & Ohio Railroad, was designated Maryland No. 2 and a part of the sample was used in briquetting tests 191, 192, 193, and 231.

In tests 191, 192, 193, and 231 the size of coal used was as follows: Over one-fourth inch, 2.2 per cent; one-tenth inch to one-fourth inch, 9.8 per cent; one-twentieth inch to one-tenth inch, 26 per cent; one-fortieth inch to one-twentieth inch, 28.2 per cent; through one-fortieth inch, 33.8 per cent. Various proportions of binder (5, 6, and 7 per cent) made equally good briquets on the English machine. The briquets had smooth, firm surfaces, characteristic glossy fracture, and sharp edges. Renfrow briquets were satisfactory, but owing to faulty operation in the heater the best results were not obtained. Excellent briquets can be made from this coal with 5 per cent of binder.

Briquetting tests.

	Test No.			
	191.	192.	193.	231.
Details of manufacture:				
Machine used.....	English.	English.	English.	Renfrow No.1.
Briquetting temperature.....°F.	185	185	185	131
Binder—				
Kind.....	W. G. P.	W. G. P.	W. G. P.	W. G. P.
Laboratory No.....	4543	4543	4543	4806
Amount.....per cent.	5	6	7	8
Weight of—				
Fuel briquetted.....pounds..	4,000	4,000	4,000	17,000
Briquets, average.....do....	3.35	3.32	3.47	0.453
Drop test (1-inch screen):				
Held.....per cent.....	75.9	79.3	82.5	24.0
Passed.....do.....	21.1	20.7	17.5	76.0
Tumbler test (1-inch screen):				
Held.....do.....	74.0	78.2	80.2	66.5
Passed (fines).....do.....	26.0	21.8	19.8	33.5
Fines through 10-mesh sieve.....do....	75.6	77.8	63.0	85.0
Weathering test:				
Time exposed.....days..	5	5	5	
Condition.....	A	A	A	
Absorption test:				
Time immersed.....days..	19	19	19	19
Water absorbed.....per cent..	11.9	15.6	10.5	14.4
Average for first 4 days.....do....	1.45	1.85	1.30	2.25
Specific gravity (apparent).....	1.118	1.076	1.143	1.071

Chemical analyses of coal and briquets.

	Raw fuel.	Tests 191, 192, and 198.	Test 231.
Laboratory No.....	4335	4415	4767
Proximate:			
Moisture..... per cent..	2.35	6.83	4.21
Volatile matter..... do.	16.97	19.71	20.71
Fixed carbon..... do.	72.53	66.01	68.54
Ash..... do.	8.15	7.45	6.54
Sulphur..... do.	.76	.86	.94
Ultimate:			
Hydrogen..... do.	4.44	4.33	4.61
Carbon..... do.	82.18	81.52	83.26
Nitrogen..... do.	1.48	1.79	1.64
Oxygen..... do.	2.77	3.44	2.68

Extraction analyses.

	Pitches.		Fuel.	Briquets.	
			Mary- land No. 2.	Tests 191, 192, and 193.	Test 231.
Laboratory No.....	4546	4906	4335	4415	4767
Air-drying loss..... per cent..			1.80	6.00	3.40
Extracted by CS ₂ (as received)..... do.	96.66	96.90	.20	5.42	7.61
Pitch in briquet by analysis..... do.				5.25	7.69
Heat value..... B. t. u.	16,960	16,864	14,173	13,581	14,008

MISSOURI NO. 1.

A sample of bituminous run-of-mine coal from New Home, Bates County, on the Frisco system, was designated Missouri No. 1, and a part of the sample was subjected to briquetting tests on the English machine at the St. Louis plant.

At the time this coal was tested only the hard pitch was available. Two and a half tons of this coal that had previously been washed was briquetted with 11.5 per cent of pitch A. The briquets were black in color and sufficiently hard to stand rough handling, but on account of the pitch setting too quickly they were insufficiently pressed and were granular and porous. This coal will, however, briquet very readily, and with softer pitches, as pitch D and pitch H, will make a good briquet with about 6 to 7 per cent of binder.

The briquets were tested under the boiler.

Chemical analyses of coal and briquets.

	Car sample.	Briquets.
Laboratory No.....	1126	1206
Proximate:		
Air-drying loss..... per cent..	5.00	2.90
Moisture..... do.	8.33	6.38
Volatile matter..... do.	23.58	37.60
Fixed carbon..... do.	28.73	41.85
Ash..... do.	19.36	14.17
Sulphur..... do.	5.25	4.56
Ultimate:		
Hydrogen..... do.	4.41	4.63
Carbon..... do.	82.18	68.52
Nitrogen..... do.	1.08	1.16
Oxygen..... do.	5.53	5.69
Heat value..... B. t. u.	10,586	11,967

MISSOURI NO. 10.

Bituminous screenings, through a $\frac{3}{4}$ -inch shaking screen, from the Bevier bed at Bevier, Macon County, on the Chicago, Burlington & Quincy Railroad, were designated Missouri No. 10, and a part of the sample was used in briquetting tests 178, 179, 241, 245, and 246.

In tests 178, 179, 241, 245, and 246 the size of coal used was as follows: Over one-fourth inch, 3 per cent; one-tenth inch to one-fourth inch, 10.2 per cent; one-twentieth inch to one-tenth inch, 17.2 per cent; one-fortieth inch to one-twentieth inch, 21.2 per cent; through one-fortieth inch, 48.4 per cent. Equally satisfactory briquets were made with 7 and 8 per cent of binder on the English machine, and 8 and 9 per cent on the Renfrow machine. The large amount of clay present made the briquets soft when warm, but very hard when cold, if the fuel was worked with high moisture content. Tests 178 and 179 were made of coal from the same car and under greater pressure than in other tests. The briquets from the two tests were harder, more cohesive, and made less slack in handling. All the briquets had rough surfaces and broke with ragged fracture. They were of a dull-gray color, owing to the amount of clay present.

Briquetting tests.

	Test No.				
	178.	179.	241.	245.	246.
Details of manufacture:					
Machine used.....	Renfrow No. 1.	English.	Renfrow No. 1.	Renfrow No. 1.	English.
Briquetting temperature.....°F..	176	185	149	149	149
Binder—					
Kind.....	W. G. P.	W. G. P.	W. G. P.	W. G. P.	W. G. P.
Laboratory No.....	4543	4543	4806	4879	4879
Amount, per cent..	8	7	9	8.5	8
Weight of—					
Fuel briquetted, pounds.....	42,000	5,000	66,000	176,000	10,000
Briquets, average, pounds.....	0.476	4.43	0.481	0.523	3.91
Drop test (1-inch screen):					
Held.....per cent..	80.5	63.3	73.0	58.0	64.4
Passed.....do.....	19.5	36.7	27.0	42.0	35.6
Tumbler test (1-inch screen):					
Held.....per cent..	94.0	90.9	91.5	87.0	64.2
Passed (fines).....do.....	6.0	9.1	8.5	13.0	35.8
Fines through 10-mesh sieve.....per cent..	99.0	80.9	95.0	90.3	50.0
Weathering test:					
Time exposed....days..	19	19			
Condition.....	B	C			
Absorption test:					
Time immersed, days..	19	19	13	13	13
Water absorbed, per cent.....	14.5	7.6	9.0	13.1	6.6
Average for first 5 days, per cent.....	1.94		1.54	2.24	1.12
Average for first 2 days, per cent.....		2.2			
Specific gravity (apparent).	1.172	1.238	1.207	1.108	1.248

Chemical analyses of coal and briquets.

	Raw fuel.	Raw fuel.	Test 178.	Test 179.	Test 241.	Test 245.	Test 246.
Laboratory No.....	4257	4803	4515	4362	4898	4876	4908
Proximate:							
Moisture..... per cent..	15.23	14.10	11.03	4.51	8.91	7.79	7.75
Volatile matter..... do.	26.32	27.64	32.24	35.53	31.51	31.46	32.48
Fixed carbon..... do.	37.95	37.02	38.67	41.37	38.17	39.87	40.95
Ash..... do.	20.50	21.24	18.06	18.69	21.11	20.88	18.82
Sulphur..... do.	3.09	4.56	3.72	3.90	4.49	4.06	4.53
Ultimate:							
Hydrogen..... do.	3.93	3.88		4.34	4.16	4.12	4.24
Carbon..... do.	58.28	56.06		64.29	60.16	60.63	62.77
Nitrogen..... do.	.99	1.01		.91	.84	.84	.91
Oxygen..... do.	8.27	9.01		6.91	6.73	6.71	6.76

Extraction analyses.

	Fuel.		Briquets.		
			Test 178.	Test 179.	Test 241.
Laboratory No.....	4257	4803	4515	4362	4898
Air-drying loss..... per cent..	13.70	10.30	9.40	1.30	
Extracted by CS ₂ (as received)..... do.	0.47	0.47	7.44	7.54	8.44
Pitch in briquet by analysis..... do.			7.03	7.13	8.27
Heat value..... B. t. u.	9,099	9,081		11,128	10,082

	Pitch.			Briquets.	
				Test 245.	Test 246.
Laboratory No.....	4543	4806	4879	4876	4908
Extracted by CS ₂ (as received)..... per cent..	99.66	96.90	94.50	8.68	7.63
Pitch in briquet by analysis..... do.				8.73	7.62
Heat value..... B. t. u.	16,999	16,864	16,805	10,262	10,580

MONTANA NO. 1.

A sample of black lignite (or subbituminous coal No. 4, nut, washed) from Red Lodge, Carbon County, on the Northern Pacific Railroad was designated Montana No. 1 and a part of the sample was subjected to briquetting tests on the English machine at the St. Louis plant.

This coal was dried slightly before briquetting. It was tested with 16 per cent of pitch A. The full limit of pressure that could be obtained on the English machine was used, and although the briquets were well pressed, they were somewhat porous and rough on the surface. They were black, but not glossy. They stood rough handling, and weighed on the average 6.30 pounds each.

The coal was briquetted with 7 per cent of pitch B, the highest pressure obtainable being used. As received from the press, the briquets were somewhat plastic, but on cooling proved to be brittle. They weighed on an average 7 pounds each.

Inability to control the moisture content of the mixture in the English machine was probably the cause of the poor results obtained

in this test, but as the mixture had to be heated in this machine by the introduction of direct steam into the mass it was impossible to prevent the condensed steam from adding to the moisture content of the mixture, and as this class of fuels has an affinity for water the moisture was readily absorbed into the coal substance and resulted in a weak briquet.

Chemical analysis.

	Car sample.		Car sample.
Laboratory No.....	1298	Ultimate:	
Proximate:		Hydrogen.....per cent..	4.65
Air-drying loss.....per cent..	2.20	Carbon.....do....	66.42
Moisture.....do....	11.05	Nitrogen.....do....	1.50
Volatile matter.....do....	35.90	Oxygen.....do....	13.16
Fixed carbon.....do....	42.08	Heat value.....B. t. u..	10,539
Ash.....do....	10.97		
Sulphur.....do....	1.73		

NEW MEXICO NO. 1.

A sample of run-of-mine subbituminous coal from a bed 3 miles north of Gallup, McKinley County, on the Santa Fe Railroad, was designated New Mexico No. 1 and a part of the sample was subjected to briquetting tests on the English and American machines at the St. Louis plant.

This coal was first briquetted with 12 per cent of pitch A, but the briquets were unsatisfactory in every way. Five and one-half tons of this coal was briquetted with 15.8 per cent pitch X, and the briquets were seemingly good in every way and could be handled easily without crumbling. On cooling, however, they began to crumble, and did not weather well. In burning they showed a decided tendency to disintegrate.

This coal was tested on the English machine with 8 per cent of pitch D. The briquets were rather crumbly and earthy, and on cooling were covered with a white efflorescent substance. They did not contain a sufficient quantity of pitch. They weighed on an average 6.83 pounds each and had a specific gravity of 1.13. Their crushing strength was 7,120 pounds, or 248 pounds to the square inch.

As 8 per cent of pitch D did not give any excess of binder, and as the briquets were not satisfactory, another ton of this coal was tested with 10 per cent of pitch D. Although the briquets fresh from the machine were somewhat stronger than those with 8 per cent of pitch D, on standing and becoming thoroughly cool they became so like those made with 8 per cent of pitch D that the two could not be distinguished from each other. This indicated that a satisfactory briquet could not be made from this coal with this pitch, even by greatly increasing the percentage. In making these briquets the English

machine was run under ideal conditions; the pressure was constant, and the coal was sufficiently dry and not too fine and could be briquetted at a moderate temperature. The briquets showed a little more of the white efflorescent substance with 10 per cent than with the 8 per cent of pitch D, although they were rather glossy. The average weight of the briquets was 6.42 pounds each and their specific gravity was 1.02. Their crushing strength was 5,900 pounds, or 206 pounds per square inch, as compared with 248 pounds per square inch for those made with 8 per cent of pitch D. The eggettes were glossy on the surface but had a dull fracture and were not tenacious. They showed a tendency to disintegrate in burning.

The most satisfactory test was with pitch H, 1 ton of coal being briquetted with 8 per cent of pitch. The briquets were hard, strong, and clean—much better in every way than any briquets of this coal made with pitch D. They weighed on the average 6.25 pounds each and their specific gravity was 1.06. Their crushing strength was 13,050 pounds, or 454 pounds to the square inch, being nearly twice the strength of those made with 8 per cent of pitch D.

Chemical analysis.

	Car sample.		Car sample.
Laboratory No.	1278	Ultimate:	
Proximate:		Hydrogen.....per cent.	5.07
Air-drying loss.....per cent.	1.60	Carbon.....do.	72.18
Moisture.....do.	12.29	Nitrogen.....do.	1.17
Volatile matter.....do.	34.58	Oxygen.....do.	12.89
Fixed carbon.....do.	46.14	Heat value.....B. t. u.	11,263
Ash.....do.	6.99		
Sulphur.....do.	.63		

NEW MEXICO NO. 2.

A sample of subbituminous slack from 1½ miles east of Gallup, McKinley County, on the Santa Fe system, was designated New Mexico No. 2, and a part of the sample was subjected to briquetting tests on both the English and American machines at the St. Louis plant.

This subbituminous coal is almost identical with New Mexico No. 1. It was tested with 9.3, 10.5, and 12 per cent of pitch X, but in every case the briquets were crumbly and no positive results could be obtained.

The coal was next tested with pitch D as a binder, 1 ton of the coal being briquetted with 7 per cent of pitch D. The briquets were crumbly, gray in color, and had an earthy fracture. They were neither hard nor strong and were porous, although the limit of pressure was applied. There did not seem to be any difference in the briquet, whether made with a low or a high pressure. A large

amount of grayish efflorescent substance appeared on the briquets almost immediately. The briquets weighed on an average 7 pounds each. The eggettes made from this mixture, although not so porous, had the same earthy appearance, fracture, and gray color.

Another ton of this coal was briquetted with 9 per cent of pitch D; an excess of pitch, due to its partial volatilization, was indicated by the bluish color on the surfaces. On account of an excess of steam, the briquets were rather soft as received from the machine, and owing to the excess of pitch many of them stuck together. Although they were better than those made with 7 per cent of pitch D, they would not have served as commercial briquets. They weighed on an average 7.12 pounds each. The coal was very dirty and contained considerable clay, and the same difficulties were experienced in briquetting it with pitch as were encountered with the Indian Territory No. 6 coal and pitch. From the work done on this sample it has been definitely proved that such dirty lignites can not well be briquetted with any commercial percentage of pitch as a binder, unless perhaps it is a pitch made from petroleum, as pitch H.

Five tons of this coal was briquetted on the American machine by using 10.25 per cent of the Hoffman patent binder, consisting of petroleum, rosin, and quicklime. The briquets were not tough, but held together rather satisfactorily when burned under the boiler. A steaming test was made of these eggettes.

Chemical analyses of coal and briquets.

	Car sample.	Briquets.
Laboratory No.....	1307	1263
Proximate:		
Air-drying loss..... per cent..	2.90	1.00
Moisture..... do.....	10.79	6.75
Volatile matter..... do.....	33.82	37.56
Fixed carbon..... do.....	36.73	39.08
Ash..... do.....	18.66	16.61
Sulphur..... do.....	1.26	1.61
Ultimate:		
Hydrogen..... do.....	4.51	4.92
Carbon..... do.....	61.73	66.73
Nitrogen..... do.....	1.06	.87
Oxygen..... do.....	10.37	7.95
Heat value..... B. t. u.	9,907	11,412

NORTH DAKOTA NO. 1.

A sample of lignite from Lehigh, Stark County, on the Northern Pacific Railroad, was designated North Dakota No. 1, and a part of the sample was subjected to briquetting tests on the English and American machines at the St. Louis plant.

This lignite is tough and woody, and does not distintegrate readily. In the first test the lignite was not previously dried and was mixed with 10 per cent of pitch A. Although the highest possible pressure

was used, the briquets were porous and had the appearance of an incoherent mass of chips. With 12 per cent of pitch A the briquets were still porous and noncoherent, and on standing for some time exposed to the weather the grains and fragments of lignite began to slack off. Between the fragments of lignite there was an excess of pitch, but it did not seem to have any adhesion for the fragments. As the briquets were cooling, the contained steam acted upon the lignites, giving them the appearance of lumps of soft, brown loam surrounded by pitch.

The next test was made with dry material, the lignite having been previously ground to about 4-mesh fineness. One ton of this dry, fine material was mixed with 12 per cent of pitch A and run through the English machine, only one side of the disintegrator being used. There was no excess of steam and the full pressure was used, but the briquets were open. With both sides of the disintegrator in use there was a slight improvement in the briquets, but they were all porous and incoherent, and on exposure to the weather began to disintegrate. There was seemingly an excess of pitch between the flakes similar to that observed in the first ton tested. The same mixtures were tested on the American machine, and the eggettes when first received had a polished, lustrous appearance, but became rough immediately and had almost no coherence. The lignite was tested further by increasing the percentage of pitch, allowing the mixtures to remain longer in the pug mill, and increasing the pressure to the limit that could be attained on the English machine, but no briquets were obtained that had any coherence, and the results were all unsatisfactory.

The poor results obtained were probably due to the absorption of moisture by the lignite from the condensed steam used to heat the mixture, as it was impossible with the English machine to prevent an excess of moisture being present in the mixture. The excess of moisture was injurious only to fuels like lignite, which readily absorb considerable moisture and are weakened by the action of the moisture.

Chemical analysis.

	Car sample.		Car sample.
Laboratory No.	1279	Ultimate:	
Proximate:		Hydrogen per cent..	4. 15
Air-drying loss per cent..	23. 60	Carbon do.	62. 26
Moisture do.	35. 38	Nitrogen do.	. 84
Volatile matter do.	29. 59	Oxygen do.	15. 88
Fixed carbon do.	25. 68	Heat value B. t. u..	6, 923
Ash do.	9. 35		
Sulphur do.	1. 55		

NORTH DAKOTA—PITTSBURGH NO. 11.

A car of run-of-mine lignite from the Scranton mine, Scranton, Bowman County, N. Dak., was designated Pittsburgh No. 11. This lignite had a dark-brown, almost black color, but became somewhat lighter colored after drying.

Briquetting tests 312, 317, 318, 319, and 322 were made of the raw lignite at Pittsburgh on the German press, without binder. House-heating boiler tests H86 and H87 were made of briquets from briquetting test 322.

Test 312 of material dried till it held 10.6 per cent moisture furnished some fairly strong briquets during the latter half of the run when conditions had become constant. The shape of these briquets was good, their surfaces were smooth, and their edges were sharp. They could be handled without too much breakage. The best briquets were made by running the machine at moderate speed—about 60 revolutions per minute.

Test 317, in which the material was dried until it contained 11 per cent moisture and the pressure used was heavier than in test 312, failed to produce satisfactory briquets. Those formed were too weak to stand handling and crumbled to pieces soon after leaving the mold. The chemical analysis reported was made of the broken briquets.

Test 318, in which the material contained 11.7 per cent moisture and the pressure was the same as in test 317, furnished some fair briquets during the latter part of the run, but they were not strong enough to stand handling. As in test 317, pieces of briquets were analyzed.

Test 319, in which the material contained 15 per cent moisture, produced excellent briquets; they had smooth surfaces sharp edges and were strong enough to bear handling.

The conditions during test 322 differed from those during test 319 only in the moisture content of the material. The briquets were strong and well formed, with smooth, polished surfaces.

Plates XVI to XXI show how briquets from tests 312, 319, and 322 withstood exposure to the weather.

This lignite was made into good briquets without the use of a binder. Satisfactory briquets can be made by drying the ground lignite until it contains 11 per cent moisture, and better briquets can be made if the moisture content is about 15 per cent. The indications are that material containing more than 15 per cent moisture can be worked on this press and will probably furnish briquets stronger than any other lignites mentioned in this report. Unfortunately, the control of the drier at the Pittsburgh plant did not permit the material, if dried at all, to contain more than 15 per cent moisture. This defect can be remedied, however, by changing the proportions or the speed of the drier.

Data relating to the test made with this lignite are summarized in the table following:

Briquetting tests.

	Test No.				
	312.	317.	318.	319.	322.
Size as shipped.....	r. o. m.	r. o. m.	r. o. m.	r. o. m.	r. o. m.
Size as used:					
Over $\frac{1}{8}$ inch, per cent.....	2.00	3.00	1.00	3.00	1.50
$\frac{1}{8}$ to $\frac{1}{4}$ inch, per cent.....	21.00	29.50	17.60	32.00	29.00
$\frac{1}{4}$ to $\frac{3}{8}$ inch, per cent.....	28.00	25.00	32.50	29.50	29.00
$\frac{3}{8}$ to $\frac{1}{2}$ inch, per cent.....	26.00	16.50	22.00	16.50	20.00
Through $\frac{1}{2}$ inch, per cent.....	23.00	26.00	27.00	19.00	20.50
Details of manufacture:					
Machine used.....	German.	German.	German.	German.	German.
Briquetting temperature, ° F.....	103	98	101	99	102
Binder used.....	None.	None.	None.	None.	None.
Tangent of die angle.....	6/89	7/89	7/89	7/89	7/89
Steam pressure on drier, pounds.....	10	10	5	4	4
Weight of—					
Fuel briquetted, pounds.....	4,383	None. ^a	2,000	4,000	6,300
Briquets, average, pounds.....	1.00		1.00	1.03	1.07
Moisture in briquet mixture, per cent.....	10.60	11.06	11.73	14.96	12.60
Drop test (1-inch screen):					
Held, per cent.....	42			44	34
Passed, per cent.....	58			56	66
Tumbler test (1-inch screen):					
Held, per cent.....	36			43	37
Passed, per cent.....	64			57	63
Fines through 10-mesh sieve, per cent.....	65			58	59
Weather test:					
Time exposed when examined, days.....	66			66	66
Condition when examined.....	E			E	D

^a The few briquets made fell to pieces when handled.

The table following shows the changes in composition of this lignite during transportation from the mine to the plant, a period of 38 days, and during storage at the plant until used for the last briquetting test, a period of 50 days. The fourth column gives an average of five proximate analyses made on different dates, and analyses of the briquets are also given.

Chemical analyses.

	Mine sample.	Car sample.	Sample as used on last test.	Average of 5 samples.
Date taken.....	Mar. 20	Apr. 23	June 12	
Laboratory No.....	7499	7677	7942	
Proximate:				
Moisture.....per cent..	41.43	38.81	32.30	32.68
Volatile matter.....do...	23.86	25.48	18.81	26.24
Fixed carbon.....do....	28.45	27.29	40.71	33.45
Ash.....do.....	6.26	8.42	8.18	7.62
Sulphur.....do.....	.74	.97	.85	.93
Ultimate:				
Hydrogen.....do.....		4.54		
Carbon.....do.....		61.20		
Nitrogen.....do.....		.82		
Oxygen.....do.....		18.09		
Heat value.....B. t. u..	6,241	6,347	7,160	7,243

Proximate analyses of briquets.

	Briquet 312.	Briquet 317.	Briquet 318.	Briquet 319.	Briquet 322.
Laboratory No.	7807	7886	7893	7901	7944
Moisture.....per cent..	10.76	10.70	12.09	12.48	11.98
Volatile matter.....do....	38.97	38.95	38.11	38.92	36.87
Fixed carbon.....do....	40.94	40.69	39.68	39.47	41.37
Ash.....do....	9.33	9.66	10.12	9.13	9.83
Sulphur.....do....	1.22	1.20	1.27	1.20	1.21
Heat value.....B. t. u..	9,434	9,470	9,189	9,184	9,445

Extraction analyses.

	Fuel.				Briquets.	
					Test 312.	Test 322.
Laboratory No.	7499	7677	7942		7807	7944
Air-drying loss.....per cent..	36.00	27.50	22.70	20.70		
Extracted by CS ₂ (moisture free)....do....			1.71	1.76	1.67	1.68

NORTH DAKOTA—PITTSBURGH NO. 13.

The sample designated Pittsburgh No. 13 consisted of 1 car of run-of-mine lignite from the Lehigh mine, Stark County, N. Dak. It was used in making gas-producer test 184 and briquetting tests 294, 298, 299, 300, 301, 302, 303, 304, 305, 306, and 307 with the German press.

The ground and dried lignite used in test 294 contained 11 per cent moisture. It made well-formed briquets that had rather brittle edges and cracked sides. They were strong enough to endure careful handling and were the strongest lot made from this fuel.

In tests 298, 299, 300, 301, 302, 303, and 304 the moisture content and the pressure were varied in an endeavor to get the best conditions, but the few briquets made were too weak and were not kept.

In test 305 the material contained about 15 per cent moisture. Some of the briquets were strong enough to endure careful handling, but they were softer than those from test 294 and crumbled more easily. Seemingly the material was too coarse and contained too much moisture. The briquets were, however, a great improvement over those from tests 298 to 304.

Test 306 produced some fairly good briquets, there being little choice between the lot and those from test 305. Seemingly the moisture content of the dried material was too high. The briquets crumbled easily and when struck gave a dull sound instead of the sharp, metallic ring of good briquets. The briquets from tests 305 and 306 were mixed in storage, and the cohesion tests were made from a mixed sample.

The briquets made in test 307 had better forms than those from any other test of this fuel, but were not so strong as those from tests 294, 305, or 306.

This series of tests has demonstrated that this lignite can be made into briquets, but further tests are desirable to determine the best conditions. Fine grinding is a requisite. About 10 to 12 per cent moisture in the material seems desirable and a heavy pressure is necessary. The proportion of natural binder in the lignite being rather low, briquets made without adding artificial binder are likely to be weak. Probably the use of a small proportion, perhaps 2 to 4 per cent, of artificial binder and a different type of press would give better briquets.

Briquetting tests.

	Test No.				
	294.	298.	299.	300.	301.
Size as shipped.....	R. o. m.	R. o. m.	R. o. m.	R. o. m.	R. o. m.
Size as used:					
Over $\frac{1}{4}$ inch..... per cent..	0.0	1.0	0.0	0.0	0.5
$\frac{1}{4}$ to $\frac{1}{2}$ inch..... do....	27.7	21.0	20.0	20.0	12.0
$\frac{1}{2}$ to $\frac{3}{4}$ inch..... do....	40.0	31.5	36.0	36.0	28.5
$\frac{3}{4}$ to 1 inch..... do....	22.3	25.5	25.0	25.0	27.0
Through 1 inch..... do....	10.0	21.0	19.0	19.0	32.0
Details of manufacture:					
Machine used.....	German.	German.	German.	German.	German.
Briquetting temperature..... °F..	90	75	75	75	68
Binder used.....	None.	None.	None.	None.	None.
Tangent of die angle.....	5.5/89	6/89	6/89	4/89	4/89
Steam pressure on drier..... pounds..	20	20	20	20	15
Weight of—					
Fuel briquetted..... do....	2,840	None.	None. ^a	None. ^a	None. ^a
Briquets, average..... do....	0.775				
Moisture in briquet mixture..... per cent..	11.13	9.19	9.88	9.88	9.19
Drop test (1-inch screen):					
Held..... do....	25				
Passed..... do....	75				
Tumbler test (1-inch screen):					
Held..... do....	30				
Passed..... do....	70				
Fines through 10-mesh sieve..... do....	67				

	Test No.					
	302.	303.	304.	305.	306.	307.
Size as shipped.....	R. o. m.	R. o. m.	R. o. m.	R. o. m.	R. o. m.	R. o. m.
Size as used:						
Over $\frac{1}{4}$ inch..... per cent..	0.5	0.5	0.5	2.0	0.8	0.8
$\frac{1}{4}$ to $\frac{1}{2}$ inch..... do....	18.0	20.0	21.0	22.5	21.5	21.5
$\frac{1}{2}$ to $\frac{3}{4}$ inch..... do....	24.0	34.5	31.5	30.5	32.2	32.2
$\frac{3}{4}$ to 1 inch..... do....	26.0	25.5	24.5	21.5	22.5	22.5
Through 1 inch..... do....	31.5	19.5	22.5	23.5	23.0	23.0
Details of manufacture:						
Machine used.....	German.	German.	German.	German.	German.	German.
Briquetting temperature..... °F..	78	74	88	81	85	102
Binder used.....	None.	None.	None.	None.	None.	None.
Tangent of die angle.....	4/89	5/89	6/89	6/89	6/89	6/89
Steam pressure on drier..... pounds..	10	10	10	8	12	16
Weight of—						
Fuel briquetted..... do....	500	700	1,950	1,880	2,138	700
Briquet, average..... do....			0.9	0.77	0.77	0.82
Moisture in briquet mixture..... per cent..	9.81	11.02	11.56	14.80	12.12	9.29
Drop test (1-inch screen):						
Held..... do....				16.00	16.00	5.00
Passed..... do....				84.00	84.00	95.00
Tumbler test (1-inch screen):						
Held..... do....				17.00	17.00	7.00
Passed..... do....				83.00	83.00	93.00
Fines through 10-mesh sieve..... do....				60.00	60.00	60.00

^a The few briquets made were too weak to handle.

^b Briquets were scrapped, as they were too weak to handle.

The table following shows the changes in composition of this fuel during transportation from the mine to the plant, a period of 12 days, and during storage at the plant until used for the last briquetting test, a period of 30 days. The fourth column is an average of eight proximate analyses of samples used on different dates. Analyses of the briquets are also given.

Chemical analyses of coal.

	Mine sample.	Car sample.	Sample as used on last test.	Average of 8 samples.
Date taken.....	Mar. 29	Apr. 10	May 10	
Laboratory No.....	7537	7553	7752	
Proximate:				
Air-drying loss..... per cent..	37.80	34.30	28.90	28.66
Moisture..... do.....	42.04	40.22	38.95	38.08
Volatile matter..... do.....	23.40	24.86	24.44	24.86
Fixed carbon..... do.....	27.67	28.03	29.38	29.74
Ash..... do.....	6.89	6.85	7.23	7.32
Sulphur..... do.....	.68	.62	.91	.90
Ultimate:				
Hydrogen..... do.....		4.28		
Carbon..... do.....		62.66		
Nitrogen..... do.....		.70		
Oxygen..... do.....		19.86		
Extract by CS ₂ (moisture free)..... do.....			1.16	
Heat value..... B. t. u..	6,079	6,246	6,327	6,579

Chemical analyses of briquets.

	Test No.							
	294	301	302	303	304	305	306	307
Laboratory No.....	7611	7702	7703	7719	7723	7726	7755	7751
Proximate:								
Air-drying loss..... per cent..	0.60							
Moisture..... do.....	8.56	8.30	10.87	10.26	10.18	12.10	11.41	10.29
Volatile matter..... do.....	37.51	37.00	36.83	38.79	37.82	35.00	36.52	37.14
Fixed carbon..... do.....	41.23	42.80	42.71	42.35	43.00	44.10	42.55	43.93
Ash..... do.....	12.70	11.90	9.59	8.60	9.00	8.80	9.52	8.64
Sulphur..... do.....	2.24	1.90	1.58	1.12	1.21	1.35	1.40	1.02
Extract by CS ₂ (moisture free)..... do.....	1.72	1.66	1.85	1.01		1.51		1.19
Heat value..... B. t. u..	9,191	9,410	9,504	9,691	9,684	9,000	9,499	9,698

NORTH DAKOTA—PITTSBURGH NO. 15.

A sample consisting of one car of 3-inch lump lignite from the McClure mine, Vanderwalker, Ward County, N. Dak., was designated Pittsburgh No. 15.

Briquetting tests 324, 325, 326, and 330 were made of it with the German press without binder. No gas-producer or steaming tests were made.

Dried lignite containing 7.4 per cent moisture was used for tests 324 and 325. In test 324 a set of dies, used in test 323, gave the heaviest pressure the machine would stand. The material stalled the machine, the end of a stamp broke, and no briquets were made.

Another set of dies giving less pressure was used for test 325 on the same lot of material. The material would not work satisfactorily, however, with this set of dies, and after several attempts the test was stopped and the rest of the lignite in the hopper was thrown away.

The next lot of ground lignite, dried to contain 10.8 per cent moisture, was used for test 326. No briquets were made and the material came out of the press in a loose form. The lignite evidently lacked binding properties, a defect that was confirmed by extraction tests made later. Another test (330) was made of a lot of material dried to contain 11.6 per cent moisture, but no briquets were formed. Similar results were obtained in tests 326 and 330. No further test could be made on this lignite at the time.

The lignite seemingly does not contain enough binding material to form briquets on the German press. Without doubt, it can be successfully briquetted by adding a binder and using a press made to work with a binder. It is desirable to make further tests of this lignite, first on the laboratory hand press and afterwards on the English machine belonging to the Bureau of Mines.

Briquetting tests.

	Test No.			
	324.	325.	326.	330.
Size as shipped.....	3-inch lump.	3-inch lump.	3-inch lump.	3-inch lump.
Size as used:				
Over $\frac{1}{4}$ inch..... per cent..	1.5	1.5	2.0	1.0
$\frac{1}{4}$ to $\frac{1}{2}$ inch..... do.....	28.0	28.0	30.0	30.0
$\frac{1}{2}$ to $\frac{3}{4}$ inch..... do.....	30.0	30.0	29.5	28.0
$\frac{3}{4}$ to 1 inch..... do.....	20.5	20.5	18.5	19.0
Through 1 inch..... do.....	20.0	20.0	20.0	22.0
Details of manufacture:				
Machine used.....	German.	German.	German.	German.
Briquetting temperature.....°F..	93	93	93.6	111
Binder used.....	None.	None.	None.	None.
Tangent of die angle.....	7/89	6/89	6/89	6/89
Steam pressure on drier..... pounds..	5	5	4	3
Weight of—				
Fuel briquetted..... do.....	None.	None.	None.	None.
Moisture in briquet mixture..... per cent..	7.40	7.40	10.80	11.60

The changes in composition of the fuel during transportation from the mine to the plant, a period of 44 days, and during storage at the plant for 73 days, are shown below. . The figures in the fourth column represent an average of three analyses made on different dates.

Chemical analyses.

	Mine sample.	Car sample.	Sample as used on last test.	Average of 3 samples.
Date taken	Apr. 3	Apr. 17	June 29	
Laboratory No.	7587	7631	8020	
Proximate:				
Air-drying loss..... per cent..	28.90	25.60	18.30	15.83
Moisture..... do.....	36.64	34.30	29.09	27.94
Volatile matter..... do.....	22.64	23.10	27.13	27.12
Fixed carbon..... do.....	30.74	29.93	34.88	34.99
Ash..... do.....	9.98	13.07	8.90	9.95
Sulphur..... do.....	.45	.52	.56	.54
Ultimate:				
Hydrogen..... do.....		3.76		
Carbon..... do.....		56.01		
Nitrogen..... do.....		.87		
Oxygen..... do.....		17.76		
Extract by CS ₂ (moisture free).....				1.08
Heat value..... B. t. u.....	6,394	6,284	7,385	7,364

PENNSYLVANIA NO. 3.

A sample of anthracite culm from near Scranton was designated Pennsylvania No. 3 and a part of the sample was subjected to briquetting tests on both the English and American machines at the St. Louis plant.

The first test was with 90 parts of the culm, 10 parts of a West Virginia coking coal, and 12½ per cent of hard pitch A. The mixture was run through the English machine, both sides of the disintegrator being used and the briquets being pressed to the limit of the machine. On account of the hardness of the anthracite coal it did not disintegrate to any great extent and the briquets resembled concrete. They were hard and tough, and even with the inferior pitch were of good quality. They weighed on an average 7.25 pounds each. During combustion they burned like lumps of anthracite coal, making little flame. On standing exposed to the weather, the briquets did not suffer to any extent except during hard rains, when their surfaces became pitted. Seemingly this pitting was due to the wearing away of the softer pitch from the harder fragments of anthracite.

The culm was next tested with the softer pitch B, by using 84 per cent of anthracite, 9 per cent of West Virginia coking coal, and 7 per cent of pitch B. For this test the coarsest and driest part of the culm was used. The combination made fair briquets, but they would have been improved if dry steam could have been used. The briquets weighed on an average 7.64 pounds each and had a specific gravity of 1.174, whereas the specific gravity of the raw culm was 1.52. The crushing strength of the briquets was only 4,500 pounds, or 156 pounds to the square inch. The eggettes of this mixture made on the American machine were harder and firmer than the corresponding briquets. They all burned like anthracite coal, except that in starting the fire they were more easily ignited.

Another ton of the anthracite culm, which had been kiln-dried, was briquetted with 7 per cent of pitch B, without the addition of any

bituminous coal. If the steam introduced into the pug mill was wet the fresh briquets were almost too soft to handle. If, however, the steam was dry, they were firm and lustrous. Although the briquets were not so lustrous as those made with soft coal, they were clean, smooth, hard, and strong, and when dropped rang like a stone. The eggettes of this mixture made on the American machine were also very good, but no better than the briquets. The eggettes also burned like lump anthracite, with little flame or crumbling. In breaking the large briquets there was little waste. The briquets weighed on an average 7.4 pounds each, and had a specific gravity of 1.28, the specific gravity of the coal being 1.52. The crushing strength of the briquets was 17,100 pounds, or 596 pounds to the square inch, this test being made of briquets that had been exposed to the weather for over a month.

One ton of the anthracite culm also was tested with 7 per cent of pitch D as a binder, but, on account of lack of pressure, the briquets were porous and not so good as those that contained 7 per cent of pitch B. They stuck together while hot, indicating that 7 per cent of pitch D was sufficient and that an increase of pitch would not improve them. They weighed only 7.06 pounds each on an average.

A 5-ton lot of the anthracite culm was briquetted with 10 per cent of West Virginia coking coal and 11½ per cent of the Hoffman patent binder, on the American machine. The eggettes were hard, smooth, and clean, with a somewhat polished surface, and stood considerable rough handling. The eggettes were tested by burning under a boiler, the fire box of which was fitted for a bituminous coal. Although made of 79½ per cent of anthracite, they did not burn like anthracite, and their oil and rosin constituents gave a long flame to the fuel, and the boiler ran at 86 per cent of its rated capacity.

From the results of these tests it is apparent that anthracite culm can be satisfactorily briquetted either in combination with a coking coal or by using simply pitch as a binder, and that the briquets have most of the properties of lump anthracite.

Chemical analyses of coal and briquets.

	Car sample.	Briquets.
Laboratory No.	1245	1294
Proximate:		
Air-drying loss per cent..	3.40	0.10
Moisture	5.41	3.00
Volatile matter	7.02	27.62
Fixed carbon	71.79	55.00
Ash	15.78	14.38
Sulphur	.74	1.13
Ultimate:		
Hydrogen	2.64	4.12
Carbon	76.81	74.08
Nitrogen	.81	1.08
Oxygen	2.27	4.44
Heat value	12,047	12,893
	B. t. u.	

PENNSYLVANIA NO. 15.

Semibituminous run-of-mine coal from the "Miller" or "B" bed at Wehrum, Indiana County, on the Pennsylvania Railroad, was designated Pennsylvania No. 15, and a part of the sample was used in briquetting tests 176 and 184, in mixture with Rhode Island No. 1. Steaming test 467 was made of the briquets from briquetting test 176. House-boiler test 47 was made of the briquets from briquetting test 184.

In test 176 the size of coal used was as follows: Over one-fourth inch, 2.2 per cent; one-tenth inch to one-fourth inch, 6 per cent; one-twentieth inch to one-tenth inch, 12 per cent; one-fortieth inch to one-twentieth inch, 19 per cent; through one-fortieth inch, 60.8 per cent. This test, with 7 per cent of binder, gave satisfactory briquets, which were tough and easily handled without breaking when warm, but were brittle when cold. They broke with characteristic smooth, glossy fracture, hard surface, and sharp edges.

In test 184 the size of coal used was as follows: Over one-fourth inch, 0.8 per cent; one-tenth inch to one-fourth inch, 7 per cent; one-twentieth inch to one-tenth inch, 15 per cent; one-fortieth inch to one-twentieth inch, 22.2 per cent; through one-fortieth inch, 55 per cent. Pennsylvania No. 15 was mixed with an equal quantity of Rhode Island No. 1 (run of mine) in this test. Excellent briquets were made with 6.25 per cent of binder on the Renfrow machine. Although the pitch used had a low melting point, the briquets handled well from the machine and piled without sticking. The outer surface was very hard and smooth, and broke without crumbling, giving a smooth fracture and sharp edges.

Briquetting tests.

	Test No.	
	176.	184.
Details of manufacture:		
Machine used.....	Renfrow No. 1.	Renfrow No. 1.
Temperature of briquets.....°F.	185	185
Binder—		
Kind.....	W. G. P.	W. G. P.
Laboratory No.....	4543	4543
Amount.....per cent.	7	6.25
Weight of—		
Fuel briquetted.....pounds.	8,000	10,000
Briquets, average.....do.	0.420	0.5
Drop test (1-inch screen):		
Held.....per cent.	50.5	68.5
Passed.....do.	49.5	31.5
Tumbler test (1-inch screen):		
Held.....do.	70.5	93.0
Passed (fines).....do.	29.5	7.0
Fines through 10-mesh sieve.....do.	85.0	91.4
Weathering test:		
Time exposed.....days.	53	11
Condition.....	A	A
Absorption test:		
Time immersed.....days.	19	16
Water absorbed.....per cent.	22.0	13.3
Average for first 4 days.....do.	4.05	
Average for first 5 days.....do.		1.90
Specific gravity (apparent).....do.	1.043	1.275

Chemical analyses of coal and briquets.

	Fuel.		Test 176.	Test 184.
	Pa. No. 15.	R. I. No. 1.		
Laboratory No.	4104	3216		4913
Proximate:				
Moisture.....per cent..	2.57	2.41	3.90	0.74
Volatile matter.....do..	18.09	4.92	23.35	15.96
Fixed carbon.....do..	69.01	73.61	64.65	69.71
Ash.....do..	10.33	19.06	8.10	13.59
Sulphur.....do..	3.97	.07	3.11	2.61
Ultimate:				
Hydrogen.....do..	4.25	.65	4.35	3.05
Carbon.....do..	77.59	76.95	78.71	77.48
Nitrogen.....do..	1.19	.17	1.09	.49
Oxygen.....do..	2.30	2.63	4.18	2.65

Extraction analyses.

	Pitch.	Fuel.		Briquets.	
		Pa. No. 15.	R. I. No. 1.	Test 176.	Test 184.
Laboratory No.	4543	4104	3216		4913
Air-drying loss.....per cent..		2.10	2.00	2.80	0.03
Extracted by CS ₂ (as received).....do..	99.66	0.77	0.02	5.72	6.25
Pitch in briquet by analyses.....do..				5.02	5.91
Heat value.....B. t. u..	16,969	13,712	10,996	13,702	12,793

PENNSYLVANIA NO. 16.

Semibituminous run-of-mine coal from the "D" bed at Hastings, Cambria County, on the Pennsylvania Railroad, was designated Pennsylvania No. 16 and a part of the sample was used in briquetting tests 172, 173, and 174. Steaming test 468 was made of equal parts of briquets from the three briquetting tests.

In tests 172, 173, and 174 the size of coal used was as follows: Over one-fourth inch, 2.2 per cent; one-tenth inch to one-fourth inch, 5.4 per cent; one-twentieth inch to one-tenth inch, 11.2 per cent; one-fortieth inch to one-twentieth inch, 21.4 per cent; through one-fortieth inch, 59.8 per cent. There was no difference in physical appearance between English briquets with 6 and those with 7 per cent of binder. All briquets were satisfactory, with smooth, very hard surface, characteristic glossy fracture, and sharp edges. Renfrow briquets broke without crumbling and were easily transported from the machine when warm.

Briquetting tests.

	Test No.		
	172.	173.	174.
Details of manufacture:			
Machine used.....	English.	English.	Renfrow No. 1.
Temperature of briquets.....°F.	203	208	185
Binder—			
Kind.....	C. T. P.	C. T. P.	C. T. P.
Laboratory No.....	4319	4319	4319
Amount.....per cent.	6	7	7
Weight of—			
Fuel briquetted.....pounds.	3,500	3,500	5,000
Briquets, average.....do.	3.66	3.69	0.428
Drop test (1-inch screen):			
Held.....per cent.	76.1	80.4	44.5
Passed.....do.	23.9	19.6	55.5
Tumbler test (1-inch screen):			
Held.....do.	78.8	81.7	60.5
Passed (fines).....do.	21.2	18.3	30.5
Fines through 10-mesh sieve.....do.	64.0	63.9	91.0
Weathering test:			
Time exposed.....days.	54	50	54
Condition.....	A	A	B
Absorption test:			
Time immersed.....days.	19	19	19
Water absorbed.....per cent.	13.9	14.7	19.0
Average for first 4 days.....do.	1.35	1.30	3.30
Specific gravity (apparent).....	1.113	1.110	1.066

Chemical analyses of coal and briquets.

	Car sam- ple.	Tests 172, 173, and 174.*
Laboratory No.....	4169	4253
Proximate:		
Moisture.....per cent.	4.25	5.27
Volatile matter.....do.	21.79	23.69
Fixed carbon.....do.	66.09	62.69
Ash.....do.	7.87	5.35
Sulphur.....do.	1.59	1.68
Ultimate:		
Hydrogen.....do.	4.44	4.36
Carbon.....do.	79.29	79.43
Nitrogen.....do.	1.34	1.29
Oxygen.....do.	5.05	4.35

* Composite sample of briquets used in steaming test 468.

Extraction analyses.

	Pitch.	Fuel, Pa. No. 16.	Briquets, tests 172, 173, 174.
Laboratory No.....	4319	4169	4253
Air-drying loss.....per cent.		3.90	4.50
Extracted by CS ₂ (as received).....do.	66.25	.06	5.35
Pitch in briquet by analysis.....do.			8.00
Heat value.....B. t. u.	16,139	13,513	13,487

PENNSYLVANIA NO. 18.

Semibituminous run-of-mine coal from a mine working the "Miller" bed, outcropping at Lloydell, Cambria County, on the Pennsylvania Railroad, was designated Pennsylvania No. 18, and a part of the sam-

ple was used in briquetting tests 196, 197, 200, 201, 202, 205, 232, 236, and 250; in mixture with Rhode Island No. 1 in briquetting test 243; and in mixture with miscellaneous No. 9 in briquetting tests 238, 239, 240, and 248. Four cars were shipped to the fuel-testing plant at St. Louis, and one car was shipped to the Pennsylvania Railroad locomotive testing plant at Altoona, Pa. Some of the briquets made from this coal were tested as locomotive fuel at Altoona, Pa., by the Pennsylvania lines.

In tests 196, 197, 200, 201, 202, 205, 232, 236, and 250 the size of coal as used was as follows: Over one-fourth inch, 1.4 per cent; one-tenth inch to one-fourth inch, 5 per cent; one-twentieth inch to one-tenth inch, 10.6 per cent; one-fortieth inch to one-twentieth inch, 24.8 per cent; through one-fortieth inch, 58.2 per cent. Excellent briquets were made on the English machine with 5, 6, and 7 per cent of binder; 8 per cent of binder gave good results on the Renfrow machine, but those with 6 and 7 per cent of binder were not satisfactory. This coal worked equally well with high and low moisture content, making briquets with hard surfaces that were easily handled and piled. The fracture was smooth, the particles of coal being so thoroughly cemented that the appearance was like a solid mass.

Briquetting tests.

	Test No.				
	196.	197.	200.	201.	202.
Details of manufacture:					
Machine used.....	English.	English.	English.	(Renfrow No. 1.	Renfrow No. 1.
Temperature of briquets.....°F..	158	140	185	149	149
Binder—					
Kind.....	W.G.P.	W.G.P.	W.G.P.	W.G.P.	W.G.P.
Laboratory No.....	4683	4683	4683	4683	4683
Amount.....per cent..	6	7	5	6	7
Weight of—					
Fuel briquetted.....pounds..	24,000	128,000	24,000	3,600	6,500
Briquets, average.....do..	3.57	3.62	3.66	0.456	0.439
Drop test (1-inch screen):					
Held.....per cent..	70.6	66.7	62.5	21.0	41.0
Passed.....do..	29.4	33.3	37.5	79.0	59.0
Tumbler test (1-inch screen):					
Held.....do..	66.0	66.1	72.2	51.0	71.0
Passed (fines).....do..	34.0	33.9	27.8	49.0	29.0
Fines through 10-mesh sieve.....do..	76.0	59.5	77.6	89.7	89.2
Weathering test:					
Time exposed.....days..	1				
Condition.....	A				
Absorption test:					
Time immersed.....days..	19	19	13	13	13
Water absorbed.....per cent..	14.9	12.0	11.5	17.4	16.0
Average for first 4 days.....do..	1.85	1.55	2.10	3.12	2.68
Specific gravity (apparent).....	1.129	1.161	1.131	1.088	1.106

Briquetting tests—Continued.

	Test No.			
	205.	232.	238.	250.
Details of manufacture:				
Machine used.....	{Renfrow No. 1.	{Renfrow No. 1.	{English.	{English.
Temperature of briquets.....°F.	158	167	158	157
Binder—				
Kind.....	W. G. P.	W. G. P.	W. G. P.	(a)
Laboratory No.....	4806	4806	4806	4825
Amount.....per cent..	8	8	7	8
Weight of—				
Fuel briquetted.....pounds..	6,000	146,000	40,000	1,800
Briquets, average.....do....	0.407	0.469	3.62	3.91
Drop test (1-inch screen):				
Held.....per cent..	54.5	54.5	67.5	89.0
Passed.....do....	45.5	45.5	32.5	11.0
Tumbler test (1-inch screen):				
Held.....do....	85.0	89.5	61.8	91.9
Passed (fines).....do....	15.0	10.5	38.2	8.1
Fines through 10-mesh sieve.....do....	89.7	90.3	53.4	46.2
Absorption test:				
Time immersed.....days..	13	13	13	13
Water absorbed.....per cent..	15.3	11.0	11.7	5.9
Average for first 4 days.....do....	2.33	1.83	2.13	0.95
Specific gravity (apparent).....	1.102	1.164	1.156	1.240

a Wax tailings.

In test 238, with Pennsylvania No. 18, 75 per cent, and miscellaneous No. 9 (coke breeze), 25 per cent, satisfactory briquets were made with 8 per cent of binder; they were similar to briquets from Pennsylvania No. 18 alone. The surfaces were smooth and hard, the fractured surface being smooth, with sharp edges. The briquets could be handled easily while warm, and did not crush or stick together when piled warm.

In test 239, using equal parts of Pennsylvania No. 18 and miscellaneous No. 9, the briquets were softer when warm than those of test 238, but could be handled without breaking; they were very hard when cold and broke without crumbling; the fractured surface was rough but firm.

In test 240, with Pennsylvania No. 18, 25 per cent, and miscellaneous No. 9, 75 per cent, the briquets were very soft when warm and could not be handled without considerable breakage, but were hard when cold. The fractured surface was very rough, with edges that crumbled easily. This test showed the necessity of handling fuel mechanically until cold to prevent breakage.

In test 243 equal parts of Pennsylvania No. 18 and Rhode Island No. 1, both in run-of-mine form, were used. An effort was made to improve the burning qualities by increasing the melting point of the binder, but owing to the hardness of the pitch used and insufficient pressure the briquets were not satisfactory. They could not be handled when warm without many being broken, but when cold were brittle, producing considerable slack in handling. No physical tests were made.

In test 248, with Pennsylvania No. 18, 10 per cent, and miscellaneous No. 9, 90 per cent, the briquets had the characteristics of those from test 240. The breakage of warm briquets in falling from the machine and during handling with the coke fork was fully 50 per cent, showing the necessity of carefully handling the briquets with a belt conveyor until they are cool. The cold briquets were handled satisfactorily.

Briquetting tests.

	Test No.				
	238.	239.	240.	243.	248.
Size as used:					
Over $\frac{1}{4}$ inch.....per cent..	1.0	0.6	1.4	1.0	1.4
$\frac{1}{4}$ to $\frac{1}{2}$ inch.....do.....	6.8	6.2	7.8	6.8	8.4
$\frac{1}{2}$ to $\frac{3}{4}$ inch.....do.....	14.2	15.0	16.6	17.6	16.8
$\frac{3}{4}$ to 1 inch.....do.....	21.8	23.4	26.2	25.2	29.0
Under $\frac{1}{4}$ inch.....do.....	56.2	54.8	48.0	49.4	44.4
Details of manufacture:					
Machine used.....	{Renfrow	Renfrow	Renfrow	Renfrow	Renfrow
	No. 1.	No. 1.	No. 1.	No. 1.	No. 1.
Temperature of briquets.....°F.	167	167	167	185	158
Binder—					
Kind.....	W.G.P.	W.G.P.	W.G.P.	W.G.P.	W.G.P.
Laboratory No.....	4806	4806	4806	4825	4579
Amount.....per cent..	8.0	8.0	8.0	8.0	8.0
Weight of—					
Fuel briquetted.....pounds..	10,000	8,000	8,000	2,000	2,000
Briquets, average.....do.....	0.485	0.479	0.568		0.479
Drop test (1-inch screen):					
Held.....per cent.....	48.0	56.5	40.0		61.0
Passed.....do.....	52.0	43.5	60.0		39.0
Tumbler test (1-inch screen):					
Held.....do.....	87.0	78.5	90.0		80.0
Passed (fines).....do.....	13.0	21.5	10.0		20.0
Fines through 10-mesh sieve.....do.....	84.4	92.6	100.0		85.2
Absorption test:					
Time immersed.....days.....	8	10	9		23
Water absorbed.....per cent..	13.9	15.8	14.0		19.5
Average for first 7 days.....do.....		2.07			
Average for first 5 days.....do.....	2.42		2.4		2.68
Specific gravity (apparent).....	1.165	1.177	1.280		1.148

Chemical analyses of coal and briquets.

	Car sample.	Raw fuel.	Test 196.	Test 197.	Test 200.
Laboratory No.....	4509	4490	5061	4755	5052
Proximate:					
Air-drying loss.....per cent..	4.10	8.10		5.10	
Moisture.....do.....	4.46	8.72	3.25	5.95	3.01
Volatile matter.....do.....	15.44		18.17	18.41	17.84
Fixed carbon.....do.....	71.63		69.87	67.00	70.53
Ash.....do.....	8.47		8.71	8.64	8.62
Sulphur.....do.....	1.49		1.42	1.34	1.38
Ultimate:					
Hydrogen.....do.....	4.50		4.12	4.16	4.13
Carbon.....do.....	81.06		81.41	80.41	82.27
Nitrogen.....do.....	1.34		1.19	1.19	1.27
Oxygen.....do.....	2.68		2.81	3.63	2.02
Extract by CS ₂ (as received).....do.....		.69	5.44	5.61	4.64
Pitch in briquet by analysis.....do.....			5.36	5.56	4.51
Heat value.....B. t. u.....	13,682		14,029	13,442	13,946

Chemical analyses of coal and briquets—Continued.

	Tests 201 and 202.	Tests 205 and 232.	Test 236.	Test 238.	Test 239.
Laboratory No.	4512	5049	4797	4930	4932
Proximate:					
Air-drying loss. per cent.	3.90		3.80	1.00	0.90
Moisture. do.	4.05	2.41	4.71	1.83	1.30
Volatile matter. do.	19.68	19.41	18.43	15.05	13.73
Fixed carbon. do.	68.56	68.86	69.14	69.52	69.67
Ash. do.	9.71	9.32	7.72	13.60	13.50
Sulphur. do.	1.72	1.75	1.19	1.61	1.19
Ultimate:					
Hydrogen. do.	4.29	4.10	4.40	3.12	2.81
Carbon. do.	79.78	81.20	81.77	77.24	70.94
Nitrogen. do.	1.05	1.11	1.26	.70	.75
Oxygen. do.	2.99	2.25	3.22	3.48	8.79
Extract by CS ₂ (as received). do.	5.73	8.11	7.38	7.21	7.52
Pitch in briquet by analysis. do.	5.70	7.78	6.98	7.17	7.35
Heat value. B. t. u.	13,547	13,937	13,873	13,185	12,787

	Test 240.	Test 243.	Test 250.	Rhode Island No. 1.	Miscel- laneous No. 9.
Laboratory No.	4977	4832	4906	3216	4763
Proximate:					
Moisture. per cent.	1.06	1.34	2.55	2.41	1.95
Volatile matter. do.	15.87	16.39	24.23	4.92	.89
Fixed carbon. do.	69.24	70.34	62.80	73.61	75.50
Ash. do.	13.83	11.93	10.42	19.06	21.66
Sulphur. do.	1.44	1.37	1.99	.07	.66
Ultimate:					
Hydrogen. do.	2.81	3.46	4.36	.65	
Carbon. do.	76.02	77.79	78.64	76.95	
Nitrogen. do.	.81	.53	.88	.17	
Oxygen. do.	4.92	4.74	3.39	2.63	

Extraction analyses.

	Pitches.			Fuel.		Briquets.		
				Rhode Island No. 1.	Miscel- laneous No. 9.	Test 240.	Test 243.	Test 250.
Laboratory No.	4625	4683	4806	3216	4763	4977	4832	4906
Air-drying loss. per cent.				2.00	1.30		0.70	1.70
Extract by CS ₂ (as received) per cent.	90.56	89.31	96.90	.02	.15	7.43	8.84	7.96
Pitch in briquet by analy- sis. per cent.						7.14	9.41	7.33
Heat value. B. t. u.	16,576	16,637	16,864	10,996	10,870	12,721	13,387	13,690

PENNSYLVANIA NO. 19.

Bituminous run-of-mine coal from the Pittsburgh bed one-fourth mile north of Herminie, Westmoreland County, on the Pennsylvania Railroad, was designated Pennsylvania No. 19, and a part of the sample was used in briquetting tests 218, 219, and 242. Steaming test 508 was made of equal quantities of briquets from briquetting tests 218 and 219.

In tests 218, 219, and 242 the size of coal used was as follows: Over one-fourth inch, 3 per cent; one-tenth inch to one-fourth inch, 8.8 per cent; one-twentieth inch to one-tenth inch, 17 per cent; one-fortieth inch to one-twentieth inch, 26.3 per cent; through one-fortieth inch, 47.6 per cent.

Briquets with 6, 7, and 8 per cent of binder appeared equally satisfactory. Their edges were sharp, their surfaces hard and smooth, and their fracture clear with firm edges. Renfrow briquets had a glossy surface. All briquets stood handling and piling well.

Briquetting tests.

	Test No.		
	218.	219.	242.
Details of manufacture:			
Machine used.....	English.	English.	Renfrow No. 1.
Temperature of briquets°F.	185	185	149
Binder—			
Kind.....	W. G. P.	W. G. P.	W. G. P.
Laboratory No.....	4683	4683	4906
Amount..... per cent.	6	7	8
Weight of—			
Fuel briquetted..... pounds.	8,000	4,000	4,000
Briquets, average..... do.	3.65	3.74	0.431
Drop test (1-inch screen):			
Held..... per cent.	73.8	78.5	78.5
Passed..... do.	26.2	21.5	21.5
Tumbler test (1-inch screen):			
Held..... do.	75.6	78.8	91.5
Passed (fines)..... do.	24.4	21.2	8.5
Fines through 10-mesh sieve..... do.	70.0	65.0	90.09
Absorption test:			
Time immersed..... days.	13	13	10
Water absorbed..... per cent.	8.6	8.4	7.9
Average for first 8 days..... do.	1.0	1.0	1.0
Specific gravity (apparent).....	1.124	1.133	1.125

Chemical analyses of coal and briquets.

	Car sample.	Tests 218 and 219.	Test 242.
Laboratory No.....	4489	4648	4828
Proximate:			
Moisture..... per cent.	3.39	5.12	1.75
Volatile matter..... do.	31.79	33.00	33.50
Fixed carbon..... do.	56.46	54.36	55.24
Ash..... per cent.	8.36	7.52	9.51
Sulphur..... do.	1.05	1.05	1.36
Ultimate:			
Hydrogen..... do.	4.85	4.76	4.52
Carbon..... do.	77.03	77.93	75.91
Nitrogen..... do.	1.44	1.41	1.35
Oxygen..... do.	6.94	6.86	7.16

Extraction analyses.

	Pitches.		Fuel.	Briquets.	
	4683	4806	Pa. No. 19.	Tests 218 and 219.	Test 242.
Laboratory No.....	4683	4806	4489	4648	4828
Air-drying loss..... per cent.			2.40	4.00	0.50
Extracted by CS ₂ (as received)..... do.	89.31	96.90		5.80	9.16
Pitch in briquet by analysis..... do.				5.71	8.21
Heat value..... B. t. u.	16,637	16,864	13,699	13,540	13,646

PENNSYLVANIA NO. 20.

Semibituminous run-of-mine coal from the Lower Kittanning or B bed, 1½ miles east of Seward, Westmoreland County, on the Pennsylvania Railroad, was designated Pennsylvania No. 20, and a part of the sample was used in briquetting tests 198, 208, 209, 212, 213, 215, and 216. Steaming test 512 was made with equal weights of briquets from briquetting tests 215 and 216. Steaming test 514 was made with equal weights of briquets from tests 208 and 209.

In tests 198, 208, and 209 the size of coal used was as follows: Over one-fourth inch, 0.8 per cent; one-tenth inch to one-fourth inch, 3.6 per cent; one-twentieth inch to one-tenth inch, 11.2 per cent; one-fortieth inch to one-twentieth inch, 27 per cent; through one-fortieth inch, 57.4 per cent. Briquets from both machines had similar appearance, with smooth, hard surface; were very brittle and broke with a glossy fracture and sharp edges. The percentage of binder seemed to have little effect on their brittleness, although Renfrow briquets, with 8 per cent of binder, handled with less breakage.

In tests 212, 213, 215, and 216 washed coal was used in sizes as follows: Over one-fourth inch, 0.8 per cent; one-tenth inch to one-fourth inch, 4.8 per cent; one-twentieth inch to one-tenth inch, 16 per cent; one-fortieth inch to one-twentieth inch, 26 per cent; through one-fortieth inch, 52.4 per cent. There was no noticeable difference between the briquets from this test and those made from raw coal.

Briquetting tests.

	Test No.						
	198.	208.	209.	212.	213.	215.	216.
Details of manufacture:							
Machine used.....	English.	Renfrow No. 1.	Renfrow No. 1.	Renfrow No. 1.	Renfrow No. 1.	English.	English.
Temperature of briquets, °F.	158	158	158	158	158	176	176
Binder—							
Kind.....	W. G. P.	W. G. P.	W. G. P.	W. G. P.	W. G. P.	W. G. P.	W. G. P.
Laboratory No.....	4683	4683	4683	4683	4683	4683	4683
Amount, per cent.....	6	7	8	7	8	6	7
Weight of—							
Fuel briquetted, pounds	3,200	4,500	8,000	6,500	6,500	6,400	3,300
Briquets, average, pounds.....	3.52	0.451	0.457	0.427	0.458	3.63	3.44
Drop test (1-inch screen):							
Held, per cent.....	74.8	19.5	26.0	23.0	19.5	74.6	71.9
Passed, per cent.....	25.2	80.5	74.0	77.0	80.5	25.4	28.1
Tumbler test (1-inch screen):							
Held, per cent.....	71.0	54.0	61.5	67.0	64.0	74.0	70.0
Passed (fines), per cent.....	29.0	46.0	38.5	33.0	36.0	26.0	29.5
Fines through 10-mesh sieve, per cent.....	65.4	86.8	86.4	91.6	87.3	63.2	70.4
Absorption test:							
Time immersed, days.....	13	13	13	13	13	13	13
Water absorbed, per cent....	14.5	15.5	15.5	19.0	14.5	9.5	11.0
Average for first 5 days, per cent.....	2.34	2.78	2.66	3.1	2.50	1.56	1.56
Specific gravity (apparent).....	1.141	1.11	1.127	1.043	1.144	1.148	1.121

Chemical analyses of coal and briquets.

	Raw fuel.	Washed fuel.	Test 198.	Tests 208 and 209.	Test 212.	Test 213.	Tests 215 and 216.
Laboratory No.	4517	4498	4769	4713	4726	4885	4726
Proximate:							
Moisture..... per cent.	4.00	3.98	6.16	2.79	3.50	1.23	3.50
Volatile matter..... do.	15.89	28.13	19.23	21.11	19.98	20.58	19.98
Fixed carbon..... do.	69.57	57.73	64.38	67.79	67.71	67.74	67.71
Ash..... do.	10.54	10.16	10.28	8.51	8.81	10.45	8.81
Sulphur..... do.	2.85	1.00	2.68	1.91	1.59	2.98	1.59
Ultimate:							
Hydrogen..... do.	4.53	4.71	4.20	4.42	4.39	4.56	4.39
Carbon..... do.	78.55	76.38	78.12	80.25	81.06	79.21	81.06
Nitrogen..... do.	1.17	1.59	1.09	1.09	1.06	1.12	1.06
Oxygen..... do.	1.90	5.70	2.83	3.73	2.72	1.51	2.72

Extraction analyses.

	Pitch.	Fuel.		Briquets.				
				Test 198.	Tests 208 and 209.	Test 212.	Test 213.	Tests 215 and 216.
Laboratory No.	4683	4517	4498	4769	4713	4726	4885	4726
Air-drying loss..... per cent.		3.60	3.10	5.50	2.00	3.10	0.60	3.10
Extract by CS ₂ (as received)..... do.	89.31		0.99	5.29	6.72	6.61	7.98	6.61
Pitch in briquet by analysis..... do.				5.00	6.49	6.87	7.92	6.87
Heat value..... B. t. u.	16,637	13,347	13,311	13,198	13,981	13,988	13,896	13,988

PENNSYLVANIA NO. 22.

Bituminous run-of-mine coal from the Pittsburgh bed at Huff, Westmoreland County, on the Pennsylvania Railroad, was designated Pennsylvania No. 22 and a part of the sample was used in briquetting tests 211, 222, and 223. Steaming test 510 was made of equal weights of briquets from briquetting tests 222 and 223.

In tests 211, 222, and 223 the size of coal used was as follows: Over one-fourth inch, 1.6 per cent; one-tenth inch to one-fourth inch, 17.8 per cent; one-twentieth inch to one-tenth inch, 33.4 per cent; one-fortieth inch to one-twentieth inch, 23.2 per cent; through one-fortieth inch, 24 per cent. English briquets with 7 per cent of binder showed better outer surface than those with 6 per cent of binder. The surfaces of all the briquets were hard, clean, sharp, and glossy, with characteristic fracture. Renfrow briquets dropped from the conveyor belt without breakage.

Briquetting tests.

	Test No.		
	211.	222.	223.
Details of manufacture:			
Machine used.....	Renfrow No.1.	English.	English.
Temperature of briquets.....°F.	158	185	158
Binder—			
Kind.....	W. G. P.	W. G. P.	W. G. P.
Laboratory No.....	4683	4683	4683
Amount.....per cent.	7	7	6
Weight of—			
Fuel briquetted.....pounds.	4,000	19,000	3,200
Briquets, average.....do.	0.442	3.87	3.93
Drop test (1-inch screen):			
Held.....per cent.	54.5	79.8	62.2
Passed.....do.	45.5	20.2	37.8
Tumbler test (1-inch screen):			
Held.....do.	85.0	73.7	67.8
Passed (fines).....do.	15.0	26.3	32.2
Fines through 10-mesh sieve.....do.	90.8	65.0	70.0
Absorption test:			
Time immersed.....days.	13	13	13
Water absorbed.....per cent.	11.7	9.6	8.9
Average for first 5 days.....do.	1.78	1.48	1.38
Specific gravity (apparent).....	1.110	1.160	1.164

Chemical analyses of coal and briquets.

	Raw fuel.	Test 211.	Test 222.	Test 223.
Laboratory No.....	4498	4684	4704	4706
Proximate:				
Moisture.....per cent.	3.98	2.21	5.15	3.55
Volatile matter.....do.	28.13	32.11	29.60	30.57
Fixed carbon.....do.	57.73	55.12	55.08	54.27
Ash.....do.	10.16	10.56	10.17	11.61
Sulphur.....do.	1.00	1.13	1.17	1.07
Ultimate:				
Hydrogen.....do.	4.71	4.53	4.46	3.55
Carbon.....do.	76.38	75.71	75.40	75.94
Nitrogen.....do.	1.59	1.39	1.39	1.24
Oxygen.....do.	5.70	6.41	6.80	6.12

Extraction analyses.

	Pitch.	Fuel.	Briquets.		
		Pa. No. 22.	Test 211.	Test 222.	Test 223.
Laboratory No.....	4683	4498	4684	4704	4706
Air-drying loss.....per cent.		3.10	1.10	4.00	2.30
Extract by CS ₂ (as received).....do.	89.31	.76	6.77	5.45	6.20
Pitch in briquet by analysis.....do.			6.78	5.30	6.09
Heat value.....B. t. u.	16,637	13,311	13,561	13,183	13,275

PENNSYLVANIA—JAMESTOWN NO. 18.

The coal designated Jamestown No. 18 was anthracite coal of buckwheat size and was briquetted in a mixture with Jamestown No. 11 coal in test 290. The data obtained from this test are presented in the section entitled "West Virginia—Jamestown No. 11."

PHILIPPINE ISLANDS—PITTSBURGH NO. 112.

A sample of 5 tons of run-of-mine subbituminous from Batan Island, one of the Philippine Islands, was furnished by the Bureau of Insular Affairs, War Department, for tests and was designated Pittsburgh No. 112. A part of the sample was tested on the hydraulic-laboratory press and later two series of tests of the coal were made under the writer's supervision at the plant of the Indianapolis Pressed Fuel Co., Indianapolis, Ind., as the English machine of the briquetting section was not in condition to make these tests and as the use of the commercial plant was kindly offered by G. W. Ladley, secretary and treasurer of the company.

The fuel has a structure similar to bituminous coal and yet the analysis would indicate that it is a lignite, although very near the subbituminous classification, the carbon-hydrogen ratio of the coal being 10: .66. This coal is very weak and easily crushed to the fineness desired for briquetting. On account of its structural weakness briquetting was expected greatly to improve this fuel, making it stronger, of better heating value, and of better weather-resisting qualities.

The first series of tests at the plant of the Indianapolis Pressed Fuel Co. was made of fuel that had been dried and crushed at the Pittsburgh plant of the bureau. The second series of tests at the Indianapolis plant was made of fuel that had been crushed at the Pittsburgh plant of the bureau but had not been dried. Two varieties of tests were made in each series, namely: (1) Tests (331 to 346) in which the material was mixed and run through the Indianapolis plant in the regular manner. (2) Tests (347 to 353) in which the material was mixed, heated in small lots, and placed in molds of the Ladley press by hand, being pressed in the usual manner.

The tests made by the first method were not so successful as those made by the second method in which the mixture of fuel and binder was heated in small lots and placed in the molds by hand, as the machine was adjusted and proportioned for pressing bituminous coal. Therefore the length of travel of the plungers was not sufficient properly to compress a spongy material like the Philippine lignite when the molds were filled in the usual manner. This result should not be considered a fault of the machine, as with a different cam the same machine could be adapted to compress a lignite properly.

A description of the Ladley press used for these tests is given on page 36.

The small quantity of fuel available for these tests (only 5 tons) did not allow as wide a range of tests on the Ladley press as could be desired, but the results obtained were sufficient to determine that this coal can be made into excellent briquets with some of the binder used in the tests.

The relative weather-resisting qualities of the different briquets are shown by the results of the absorption test, in which the briquets were immersed in water and the percentage of water absorbed determined daily. Briquets that do not go to pieces quickly in the absorption test will stand the weather well.

The amount of moisture, the size of coal used, and the numbers of tests made on each of the two lots is shown below:

Tests on subbituminous coal from Batan Island, P. I.

Series of tests.	First.	Second.	
		Lot A.	Lot B.*
Test Nos.....	331-341	342-345	346-353
Percentage of moisture in fuel.....	7.22	17.99	17.99
Size as used:			
Over $\frac{1}{8}$ inch.....per cent..	1.0	11.2	1.0
$\frac{1}{8}$ to $\frac{1}{4}$ inch.....do....	14.0	29.0	14.0
$\frac{1}{4}$ to $\frac{3}{8}$ inch.....do....	30.0	29.3	30.0
$\frac{3}{8}$ to $\frac{1}{2}$ inch.....do....	27.7	17.5	27.7
Through $\frac{1}{2}$ inch.....do....	27.3	13.7	27.3

* Same sample as lot A, but lot B was screened through a $\frac{1}{4}$ -inch screen to remove large pieces.

The data obtained at the Indianapolis tests are given in the following table:

Briquetting tests.

	Test No.				
	331.	332.	333.	334.	335.
Details of manufacture:					
Machine used.....	Ladley.	Ladley.	Ladley.	Ladley.	Ladley.
Briquetting temperature.....°F..	186	184	188	185	186
Binder.....					
Kind.....	(e)	(e)	Cell pitch.	C. T. P.	C. T. P.
Laboratory No.....	34	34	9298	12009	12009
Amount.....per cent..	6	10	10	6	8
Weight of.....					
Fuel briquetted.....pounds..	200	400	150	400	400
Briquet, average.....ounces..	8	8	7.5	6.5	6.5
Moisture in briquet mixture.....per cent..	14.08	13.65	12.50	14.76	20.29
Drop test (1-inch screen):					
Held.....do....	0.0	56	26	2	24
Passed.....do....	100.0	44	74	98	76
Absorption test:					
Time immersed.....days..	0.5	0.5	0.5	20	23
Water absorbed.....per cent..	See text.	See text.	See text.	28.36	36.11
Average for first 4 days.....do....				4.97	8.33
Specific gravity (apparent).....	1.098	1.163	1.068	0.968	0.892

* Sulphite liquor.

Briquetting tests—Continued.

	Test No.				
	336.	337.	338.	339.	340.
Details of manufacture:					
Machine used.....	Ladley.	Ladley.	Ladley.	Ladley.	Ladley.
Briquetting temperature.....*F.	200	208	194	195	195
Binder—					
Kind.....	C. T. P.	C. T. P.	C. T. P.	C. T. P.	C. T. P.
Laboratory No.....	12009	12009	12009	12009	12009
Amount.....per cent.	7	6	6.5	5.5	4
Weight of—					
Fuel briquetted.....pounds.	800	500	1,600	400	200
Briquet, average.....ounces.	7	7	7	6.5	6.5
Moisture in briquet mixture.....per cent.	11.48	11.60	11.84	14.01	14.72
Drop test (1-inch screen):					
Held.....do.	6	3	15	4	(a)
Passed.....do.	94	97	85	96	
Absorption test:					
Time immersed.....days.	20	20	20	16	14
Water absorbed.....per cent.	24.96	29.53	30.28	31.02	24.71
Average for first 3 days.....do.	3.38	5.19	5.59	6.17	4.85
Specific gravity (apparent).....	1.024	0.971	0.900	0.936	0.945

	Test No.				
	341.	342.	343.	344.	345.
Details of manufacture:					
Machine used.....	Ladley.	Ladley.	Ladley.	Ladley.	Ladley.
Briquetting temperature.....*F.	200	200	201	200	176
Binder—					
Kind.....	Cell pitch.	W. G. P.	W. G. P.	Flour.	W. G. P.
Laboratory No.....	9298	12010	12010	11976	12010
Amount.....per cent.	12	6	8	8	10
Weight of—					
Fuel briquetted.....pounds.	400	200	200	250	250
Briquet, average.....ounces.	7	8	7	8	7
Moisture in briquet mixture.....per cent.	15.18	23.56	23.67	25.31	24.09
Drop test (1-inch screen):					
Held.....do.	77	0.0	12.0	43	34
Passed.....do.	23	100	88	57	66
Absorption test:					
Time immersed.....days.	0.5	5	14	0.5	12
Water absorbed.....per cent.	See text.	13.78	19.31	See text.	18.65
Average for first 4 days.....do.		4.28	4.70		5.12
Specific gravity (apparent).....	1.059	1.096	1.059	1.177	1.052

	Test No.				
	346.	347.	348.	349.	350.
Details of manufacture:					
Machine used.....	Ladley.	Ladley.	Ladley.	Ladley.	Ladley.
Briquetting temperature.....*F.	135	210	158	158	210
Binder—					
Kind.....	W. G. P.	W. G. P.	W. G. P.	W. G. P.	Flour.
Laboratory No.....	12010	12010	12010	12010	11976
Amount.....per cent.	10	6	8	10	8
Weight of—					
Fuel briquetted.....pounds.	250	200	25	20	50
Briquet, average.....ounces.	8	8.5	9	10.5	10
Moisture in briquet mixture.....per cent.	20.79	14.12	16.46	16.30	11.79
Drop test (1-inch screen):					
Held.....do.	11	(c)	(c)	32	(c)
Passed.....do.	89	(d)		68	
Absorption test:					
Time immersed.....days.	14	15	20	20	0.5
Water absorbed.....per cent.	22.02	18.97	19.16	18.49	See text.
Average for first 4 days.....do.	5.70	4.19	4.13	3.90	
Specific gravity (apparent).....	1.049	1.061	1.067	1.098	1.158

a There were not enough briquets to make this test.

b Heated over gasoline stove.

c There were not sufficient briquets for this test.

d Tests Nos. 347 to 353 were heated in small lots and filled into molds by hand.

Briquetting tests—Continued.

	Test No.		
	351.	352.	353.
Details of manufacture:			
Machine used.....	Ladley.	Ladley.	Ladley.
Briquetting temperature.....°F.	210	210	210
Binder—			
Kind.....	W. G. P.	(a)	(a)
Laboratory No.....	12010	(b)	(b)
Amount.....per cent.	10	6 and 6	5 and 5
Weight of—			
Fuel briquetted.....pounds.	20	400	100
Briquet, average.....do.	10	10	9.5
Moisture in briquet mixture.....per cent.	19.01	19.23	19.50
Drop test (1-inch screen):			
Held.....do.	(c)	94	88
Passed.....do.	6		12
Absorption test:			
Time immersed.....days.	14	3	2
Water absorbed.....per cent.	14.30	8.31	12.05
Average for first 4 days.....do.	3.63	2.77	
Specific gravity (apparent).....	1.108	1.189	1.150

a W. G. P. and flour.
b 12010 and 11976.

c There were not enough briquets to make this test.

Chemical analyses of coal and briquets.

	Raw fuel.	Raw fuel. ^a	Test 331.	Test 332.	Test 333.
Laboratory No.....	11114	11975	12037	12038	12039
Proximate:					
Air-drying loss.....per cent..	5.7	6.60	6.80	4.90	4.50
Moisture.....do.	20.08	17.99	14.08	13.65	12.50
Volatile matter.....do.	36.73	37.48	35.48	37.70	38.16
Fixed carbon.....do.	36.66	35.00	41.13	39.94	40.43
Ash.....do.	6.53	9.53	9.31	8.71	8.91
Sulphur.....do.	1.43	1.58	1.96	1.87	2.02
Ultimate:					
Hydrogen.....do.	4.67	4.40			
Carbon.....do.	66.33	62.46			
Nitrogen.....do.	1.25	1.27			
Oxygen.....do.	17.79	18.32			
Extract by CE_2 (as received).....do.	1.79				
Heat value.....B. t. u..	9,176	8,822	9,727	9,565	9,842
Specific gravity.....	1.318				

	Test No.				
	334.	335.	336.	337.	338.
Laboratory No.....	12040	11520	11518	11517	11521
Air-drying loss.....per cent..	5.90	16.30	5.40	5.20	6.20
Moisture.....do.	14.76	20.29	11.48	11.60	11.84
Volatile matter.....do.	36.35	33.13	39.85	40.65	39.95
Fixed carbon.....do.	40.36	38.34	39.60	39.92	39.53
Ash.....do.	8.53	8.24	9.07	7.83	8.68
Sulphur.....do.	1.62	1.58	1.66	1.57	1.58
Heat value.....B. t. u..	9,655	9,230	9,950	10,109	9,923

a Analysis of second lot of fuel tested at Indianapolis plant.

Chemical analyses of coal and briquets—Continued.

	Test No.				
	339.	340.	341.	342.	343.
Laboratory No.....	12041	12042	11519	11962	11963
Air-drying loss..... per cent..	4.40	5.00	9.40	16.60	14.30
Moisture..... do.....	14.01	14.72	15.18	23.56	23.67
Volatile matter..... do.....	38.67	37.97	39.70	32.43	34.30
Fixed carbon..... do.....	39.37	39.24	36.12	34.90	33.64
Ash..... do.....	7.95	8.07	9.00	9.11	8.39
Sulphur..... do.....	1.55	1.48	1.94	1.92	1.53
Heat value..... B. t. u..	9,754	9,594	9,169	8,743	8,658

	Test No.				
	344.	345.	346.	347.	348.
Laboratory No.....	11964	11966	11967	11968	11969
Air-drying loss..... per cent..	16.00	15.60	10.70	2.40	4.90
Moisture..... do.....	25.31	24.09	20.79	14.12	16.46
Volatile matter..... do.....	33.04	34.38	37.18	38.96	38.15
Fixed carbon..... do.....	33.19	33.01	33.33	37.28	36.57
Ash..... do.....	8.46	8.52	8.70	9.64	8.82
Sulphur..... do.....	1.54	1.47	1.50	1.60	1.51
Heat value..... B. t. u..	8,307	8,730	9,036	9,639	9,497

	Test No.				
	349.	350.	351.	352.	353.
Laboratory No.....	11970	11971	11972	11973	11974
Air-drying loss..... per cent..	3.60	1.00	9.50	9.30	5.40
Moisture..... do.....	16.30	11.79	19.01	19.23	19.50
Volatile matter..... do.....	38.36	41.88	39.26	40.02	38.03
Fixed carbon..... do.....	36.50	37.35	33.22	32.40	34.05
Ash..... do.....	8.84	8.98	8.51	8.35	8.42
Sulphur..... do.....	1.53	1.59	1.50	1.43	1.40
Heat value..... B. t. u..	9,531	9,293	9,241	8,822	8,930

A series of laboratory experiments, 45 in number, was also made on this fuel. The results are discussed on pages 79 to 84.

In test 331 coal and thick sulphite liquor were mixed with shovels after the liquor had been softened with steam from the steam jet. The mixture was run into the measuring machine and then through the rest of the plant. The briquets made were soft and did not have sufficient binder. Therefore 6 per cent of sulphite liquor is not sufficient for this material. When air-dried, the briquets were very soft and crumbly, the shape was spoiled, and the surfaces were badly rubbed off by transportation. In the absorption test the briquets went to pieces in water a few hours after immersion.

In test 332 fuel and sulphite liquor were mixed by hand after the binder had been melted by heating it in the water bath and applying steam from the steam jet. The materials balled considerably in the mixing. The briquets were a decided improvement on those made with 6 per cent of binder, but were not very strong when just made. It was indicated that with a proper dryer for the briquets

a satisfactory briquet can be made with this binder. After standing a day the briquets were firm and strong; hence 10 per cent of sulphite liquor was sufficient to make a satisfactory briquet of this coal; 14 per cent would be better if the briquets were not to be used immediately. In the absorption test the briquets went to pieces a few hours after immersion.

In test 333 coal and ground pitch were mixed by means of shovels, were thrown through the chute into the coal compartment of the measuring machine, and were heated by the steam heater, after which the mixture was pressed in the usual way. The briquets were weak when fresh and seemingly there was not sufficient quantity of the material (150 pounds) to fill the boxes properly. Some of the material was put into the molds by hand and compressed, making much stronger briquets. When air-dried the shape was good, the edges were fair, and the strength was fair; the surfaces rubbed off somewhat too easily. In the absorption test the briquets went to pieces a few hours after immersion.

In test 334 the fuel and pitch were mixed cold and fed into the measuring machine. When freshly made the briquets were not very strong, and it appeared that the mold boxes were not properly filled. Some of the briquets were fairly satisfactory, however, and 6 per cent of this binder may be sufficient. When air-dried the shape of the briquets was poor, the surfaces were rough and crumbly, and the edges brittle and weak. These briquets were unsatisfactory, being very weak. They resisted the water in the absorption test very well, but the amount of water absorbed indicated too much porosity and an insufficient compression. The briquets were lighter than water.

In test 335 the fuel and the pitch were mixed by hand and fed into the measuring machine. The briquets were stronger than those made with 6 per cent of binder, but 8 per cent was seemingly more than was necessary. The surfaces of the briquets were rough and there was too much moisture present. When air-dried, the shape of the briquets was poor, the surfaces were rough and crumbly, and the edges weak. The briquets were not satisfactory, being very weak. They were lighter than water and the water absorbed in the absorption test indicated too much porosity and too little compression.

In test 336 pitch and coal were mixed by hand and fed into the measuring machine, a large amount of material being used so that the mold might fill better. The machine was stopped by the action of the pitch, few briquets being obtained. The shape of the air-dried briquets was poor, the surfaces were rough and very crumbly, and the edges very weak. The briquets stood the absorption test well and should resist weathering well.

In test 337 fuel and a binder of coal-tar pitch were mixed dry and shoveled into the measuring machine. The briquets were fairly strong when fresh, and seemingly had sufficient pitch and an excess of water. The shape of the air-dried briquets was very poor, the surfaces were very rough and crumbly, and the edges rounded and crumbly. The briquets were weak and wore away very easily. The absorption test indicated that the briquets were too porous and required more compression. They were lighter than water.

In test 338 fuel and pitch were mixed by hand and fed into the measuring machine. The briquets were run out to the belt conveyor, but the fan was not working, and few of them were good as they fell off the end of the belt. A second lot was mixed and sent through with the fan running to cool the briquets, which were better than those of the previous lot. Coal and pitch were mixed for a third batch. The shape of the air-dried briquets was fairly good. Their surfaces were slightly rough and were not easily affected by rubbing, although they were somewhat rubbed as a result of shipment. The edges were slightly rounded but fairly firm. The strength of the briquets was good. They were too porous, required more compression, and were lighter than water.

In test 339 fuel and a binder of coal-tar pitch were mixed and fed into the measuring machine. This lot of briquets when fresh was seemingly as strong as those made with 6.5 per cent of binder. The shape of air-dried briquets was poor; their surfaces were rough and badly rubbed and easily crumbled; the edges were worn round, crumbled easily, and were weak. The briquets broke easily, were too porous, and lighter than water.

Test 340 furnished briquets that looked well when they came from the press but had little strength to resist handling. When air-dried they were the worst briquets of the lot. Their shape was lost, their surfaces were crumbled irregularly, and what remained of the briquets was soft and crumbly. Their edges were completely rounded away by rubbing in shipping. The briquets were very weak, were too porous, and were lighter than water.

In test 341 coal and cell pitch were mixed and sent through the machine. When air-dried, the briquets had a very good shape; the surfaces were fairly smooth and not easily rubbed; their edges, although slightly rounded, were firm, and the briquets were the strongest ones made to date from this fuel. In the absorption test the briquets went to pieces a few hours after immersion and therefore would not stand weathering well.

In test 342 coal and water-gas pitch were mixed cold and run through the heater. The mixture appeared too wet for the best results. The briquets were soft when fresh. When air-dried the briquets had a good shape; their surfaces were rough and showed the effect of

too large pieces in the mixture as the roughness did not extend over the whole surface; their edges were rounded and crumbly; the briquets were weak. The briquets resisted water well in the absorption test and absorption was complete in five days after immersion.

In test 343 coal and water-gas pitch were mixed cold and passed through the heater. The briquets appeared too wet and indicated that the amount of steaming should be reduced. When air-dried, the briquets were better than the previous lot, but their surfaces were rough and crumbly; their shape was fair; their edges were crumbly, rounded, and weak. The briquets were weak, resisted water well, but were too porous.

In test 344 flour and coal were mixed and heated in the heater before being pressed. The briquets were fairly firm when fresh, but were not very smooth on their surfaces and rubbed off too easily. When air-dried, their shape was very good, their surfaces were smooth and not easily rubbed, and their edges were sharp and rather firm. The briquets resisted water poorly and went to pieces a few hours after immersion.

In test 345 coal and pitch were mixed and fed into the middle of the heater by hand. This mixture required one-half minute to go through the heater. The briquets were fairly strong when fresh, but were rough on their surfaces and did not have enough compression. When air-dried their shape was very poor, their surfaces very rough but not easily crumbled, and their edges rough and rounded but firm. The strength of the briquets was good. The briquets possessed good water-resisting qualities but were too porous.

In test 346, to determine what effect the size of the material had on the briquets, two sacks of coal were screened through a $\frac{1}{4}$ -mesh screen. Then coal and pitch were mixed and run into the middle of the heater before being pressed. There was not much difference between these briquets and the briquets of test 345. When air-dried their shape was poor, their surfaces were rough but not easily crumbled, and their edges were rough and rounded but nevertheless were rather firm. The strength of the briquets was good. They resisted water well but would have been improved by a higher compression in the mold.

In test 347 the briquetting mixture was heated with steam from the steam jet and placed in the molds by hand. The briquets were fairly strong when fresh, but when cold they rubbed too easily and indicated too little binder. When air-dried their shape was poor, their surfaces were very rough and easily crumbled, and their edges rounded and soft. The briquets were weak. The briquets from this test resisted water well.

In test 348 the mixture was heated over a gasoline stove. The briquets had good form and surfaces but were too easily affected by

rubbing to be entirely satisfactory. When air-dried their shape was good and their surfaces were fairly smooth, but could be crumbled too easily; the edges were rounded on some of the briquets but sharp on others and were moderately firm; the strength of the briquets was fairly good, but more binder was needed. The briquets resisted water well in the absorption test.

In test 349 the briquetting mixture was heated over a gasoline stove and placed in the molds by hand. The briquets could be handled when fresh, but were not entirely satisfactory on account of the ease with which their surfaces rubbed off. When air-dried their shape was good; their surfaces were slightly rough but firm; their edges were very slightly rounded yet firm; their strength was good. The briquets resisted water well.

In test 350 the briquetting mixture was dried and heated with steam from the steam jet for one-quarter minute before being placed in the molds by hand. The briquets had an excellent appearance when fresh and were strong enough to stand handling. After having stood for one hour they did not have as sonorous a sound when struck together as those made with pitch binder. When air-dried their shape was excellent, their surfaces smooth and firm, and their edges sharp and firm. The briquets were very strong, but they did not resist the bad effects of water well, as in the absorption test they went to pieces a few hours after immersion.

In test 351 the briquetting mixture was heated with steam from the steam jet, instead of with dry heat as in test 346. After heating for one-quarter minute the mixture was placed in the molds by hand. The briquets were a decided improvement over those made with dry heat (test 346) and were strong enough to stand handling. When cold, the briquets had a satisfactory appearance and strength and did not rub to pieces as much as some of the briquets from previous tests. They had excellent water-resisting qualities and absorbed less water than those from test 346, in which less compression was obtained.

In test 352 the briquetting mixture was heated with steam from the steam jet for one-quarter minute and placed in the molds by hand. The briquets had an excellent appearance and strength when fresh and would not rub off easily. When cold, they were the best made from this fuel, having firm, hard surfaces and sharp, firm edges. They could not be broken in the hands. They stood the water-absorption test only three days, but probably would have sufficient weather-resisting qualities for commercial conditions.

In test 353 the briquetting mixture was heated one-quarter of a minute with steam from the steam jet and fed into the molds by hand. The briquets were strong enough for all purposes when fresh, and would unquestionably stand the handling incident to being placed in storage. The cold briquets had excellent appearance

and strength. Little difference was noticeable between them and the briquets made with 6 per cent of flour and 6 per cent of water-gas pitch (test 352). Either of these two briquets equals in appearance those made from Indiana coal. They stood two days' immersion in water before going to pieces, and may therefore be sufficiently weather-resisting to meet commercial conditions.

SUMMARY OF RESULTS OF TESTS OF PHILIPPINE COAL.

CRUSHING.

The Philippine coal itself is very weak and should be finely ground before briquetting (the fineness of corn meal will be sufficiently fine), as the binder, in order to be effective, must coat very small particles. Then, even if when the pressure is applied the small particles should break, the briquets would not be weakened materially.

In this respect the briquetting of this coal differs from the briquetting of anthracite and bituminous coal, for the latter are strong in themselves and all that is required of the binder is that it cement the particles of the coal together, as the individual particles are strong enough to stand the briquetting pressure without breaking.

In the case of the Philippine coal, the high pressure necessary to unite the particles by means of the binder is sufficient to crush coal particles of much size, with a consequent weak briquet; hence a liquid binder should have an especial advantage for briquetting this coal, as it would tend to be absorbed into the particles of coal, and if they should be crushed by the briquetting pressure there would be sufficient binder present to keep the briquet from becoming weak.

EFFECTS OF DRYING.

Briquets made from the undried fuel possessed greater strength than those made from the dried fuel. This difference is seemingly due to the fact that when moisture is removed from this fuel by drying, the physical strength of the fuel is decreased, and when pressure is applied to the mixture in making briquets, particles of coal are crushed, and some binders are not fluid enough to penetrate into the porous structure of the fuel, as for example, coal-tar and water-gas pitch; hence the crushed particles have no binder to cement them together and the briquets are weak.

It was noticed that the larger pieces of dried coal could be crushed in the hand, whereas pieces of the undried coal of the same size were much stronger. In tests L 95^a and L 101^a, 6 per cent of water-gas pitch was used. In test L 95 of the undried coal the briquets had a crushing strength of 950 pounds per square inch, whereas in test L 101 the crushing strength was only 600 pounds, or about two-

^a See p. 80.

thirds that of the briquet made from the undried fuel. Comparing test L 96, using 4 per cent of water-gas pitch and undried fuel, with test L 100, using 4 per cent of water-gas pitch but dried coal, it is found that the crushing strength is as 500 to 175, or nearly 3 to 1. When 8 per cent of pitch is used, the difference in strength between the briquets made from the undried fuel and the ones made from the dried fuel is not so great, the crushing strength of the former being 1,240 pounds per square inch, and that of the latter 1,100 pounds per square inch. A comparison of the briquets made with starch binder shows an even greater difference in strength between the briquets made from the undried and the dried fuel. Comparing test L 104, made with 5 per cent of starch and dried fuel, with test L 139, made with 5 per cent of starch and undried fuel, the dried-fuel briquets had a crushing strength of less than 50 pounds per square inch, whereas the undried-fuel briquets had a crushing strength of 775 pounds. With flour binder the briquets made with undried fuel were also much stronger than those made with dried fuel. Of all the briquets made, those in which flour was the binder had the greatest strength. The briquets of test 352, made with 6 per cent each of water-gas pitch and flour, were the strongest of all, and in a drop test 94 per cent of the briquets was held on a 1-inch screen after five drops. Those made by test 353 were almost as strong as those of the previous test, as they stood a drop test with 88 per cent held by the 1-inch screen. Tests 332 and 341, made respectively with 6 per cent of sulphite liquor and 12 per cent of cell pitch, furnished briquets of a satisfactory strength.

COMPRESSIBILITY.

Laboratory tests of this coal show that reduction in volume of the briquetting mixture under a pressure of 6,000 pounds per square inch varies from 35 to 45 per cent, with an average of 41.7 per cent.

BINDER REQUIRED.

As a result of these tests it was found that this fuel could not be satisfactorily briquetted without binder, even when a pressure of 30,000 pounds per square inch was used.

On account of the porosity of this fuel and its structural weakness, a larger proportion of binder is required to make it into satisfactory briquets than is required for bituminous coals.

PROPORTION OF BINDER REQUIRED.

According to laboratory briquetting tests, 7 per cent of the kind of water-gas pitch used would be sufficient to make briquets of satisfactory strength; briquets made with 8 per cent of this pitch showed an excess of binder, whereas briquets made with 6 per cent

were not quite strong enough. The tests made with the Ladley press indicate that a higher percentage of water-gas pitch would be required to make a commercial briquet—probably 10 to 12 per cent.

Tests with the Ladley machine indicated that 8 per cent of coal-tar pitch is not sufficient to produce briquets of satisfactory strength. It should be said, however, that with more compression on the machine 8 per cent of coal-tar pitch might be found to be sufficient.

Flour binder made stronger briquets from this fuel than any other one binder, but briquets made with flour binder alone did not stand the water as well as those made with pitch alone or with a compound binder of pitch and flour.

Ten per cent of sulphite liquor furnished a briquet of sufficient strength but deficient in water-resisting qualities. Twelve per cent of cell pitch produced a briquet of sufficient strength but deficient in water-resisting qualities.

More than 5 per cent of starch binder would be required to produce a briquet of satisfactory strength, and probably 8 to 10 per cent would be sufficient. None of the binders used gave satisfactory results with the dried fuel.

HEAT VALUES.

The heat values of briquets made with dried fuel were 1 to 8 per cent higher than the fuel as received at the plant, this increase of heat value being largely due to the decrease of moisture present. The briquets made from the undried fuel showed an improvement in heat value of 5 to 8 per cent when the briquets were allowed to stand for a few days after being made. The heat value of the briquets is approximately 9,500 British thermal units per pound.

RHODE ISLAND NO. 1.

Graphitic anthracite run-of-mine coal from Cranston, Providence County (near Providence), was designated Rhode Island No. 1, and a part of the sample was used in mixture with Utah No. 1 coal in briquetting test 127, with Utah No. 2 coal in briquetting test 133, with Pennsylvania No. 15 in briquetting test 184, and with Pennsylvania No. 18 in briquetting test 243. Steaming tests 414 and 415 were made of briquets from briquetting test 127, and steaming test 416 was made of briquets from briquetting test 133.

Analyses of this fuel and details of the briquetting tests are presented in the sections dealing with the coals with which this fuel was mixed as mentioned in the preceding paragraph.

RHODE ISLAND—PITTSBURGH NO. 19.

A sample of four barrels of run-of-mine graphitic anthracite coal from Cranston mine, Cranston, Providence County, was designated Pittsburgh No. 19, and briquetting test 316 was made with this

sample. The briquets made in this test were delivered for steaming tests to the company furnishing the sample of coal.

Excellent briquets were made with 6 per cent of pitch. Even less binder might have been sufficient, but the small quantity of the sample permitted making only one test on the English machine. However, laboratory tests on the hydraulic press indicated that 4 per cent of binder is sufficient to produce a strong and satisfactory briquet.

It will be noted that the briquets had a very high specific gravity owing to the high specific gravity of the coal used. The briquets obtained were very strong and difficult to fracture and had firm edges. A smoother briquet could have been made if the material used had been fine slack or more finely ground, but the sample used was too hard to be ground fine by the equipment available at the time of the test. As all of the briquets made were sent away, no drop or tumbler tests could be made of them.

Briquetting tests.

	Test 316.		Test 316.
Size as shipped.....	r. o. m.	Details of manufacture—Contd.	
Size as used:		Binder—	
Over $\frac{1}{2}$ inch..... per cent.	6.0	Kind.....	W. G. P.
$\frac{1}{2}$ inch to $\frac{3}{4}$ inch..... do.	40.0	Laboratory No.....	5940
$\frac{3}{4}$ inch to 1 inch..... do.	22.0	Amount..... per cent.	6
1 inch to $\frac{5}{8}$ inch..... do.	13.0	Weight of—	
Through $\frac{5}{8}$ inch..... do.	19.0	Fuel briquetted..... pounds.	810
Details of manufacture:		Briquets, average..... do.	4.78
Machine used.....	English.	Specific gravity (apparent).....	1.764
Briquetting temperature. °F.	162		

Chemical analyses of coal and briquets.

	Raw fuel.	Test 316.
Laboratory No.....	7790	7843
Moisture..... per cent.	4.51	2.31
Volatile matter..... do.	3.46	4.68
Fixed carbon..... do.	59.67	70.02
Ash..... do.	32.36	22.98
Sulphur..... do.	.12	.29

Extraction analyses.

	Pitch.	Fuel.	Briquets.
		R. I. No. 1.	Test 316.
Laboratory No.....	5940	7790	7843
Air-drying loss..... per cent.		4.00	
Extract by CS ₂ (as received)..... do.	96.60		
Heat value..... B. t. u.	16,780	8,411	10,247

TENNESSEE NO. 1.

Bituminous run-of-mine coal from Fork Ridge, Claiborne County, on the Louisville & Nashville Railroad, was designated Tennessee No. 1, and 18,600 pounds of the washed sample was used in briquetting test 130. Steaming tests 409 and 411 were made of briquets from briquetting test 130.

Briquets made on the English machine with 5 and 6 per cent of binder were good and firm; their edges and surfaces were harder with 6 per cent than with 5 per cent of binder. The fractures of both were sharp and clear.

Briquets made on the Renfrow machine with 6 and 6.5 per cent of binder showed dull fracture, their edges broke easily, and their surfaces were too soft. Those made with 7 per cent of binder were satisfactory, having a clear fracture and sharp edges; they broke without crumbling.

Briquetting tests.

	Test No.				
	130A.	130B.	130C.	130D.	130E.
Size as shipped.....	R. o. m.	R. o. m.	R. o. m.	R. o. m.	R. o. m.
Size as used:					
Over $\frac{1}{8}$ inch, per cent.....	2.5	2.5	2.5	2.5	2.5
$\frac{1}{8}$ inch to $\frac{1}{4}$ inch, do.....	7.7	7.7	7.7	7.7	7.7
$\frac{1}{4}$ inch to $\frac{3}{8}$ inch, per cent.....	17.8	17.8	17.8	17.8	17.8
$\frac{3}{8}$ inch to $\frac{1}{2}$ inch, per cent.....	24.1	24.1	24.1	24.1	24.1
Through $\frac{1}{2}$ inch, per cent.....	47.9	47.9	47.9	47.9	47.9
Details of manufacture:					
Machine used.....	English.	English.	Renfrow No. 1.	Renfrow No. 1.	Renfrow No. 1.
Briquetting temperature.....°F.	180	180	140	140	140
Binder—					
Kind.....	W. G. P.	W. G. P.	W. G. P.	W. G. P.	W. G. P.
Laboratory No.....	3410	3410	3410	3410	3410
Amount, per cent.....	5	6	6	6.5	7
Average weight of briquet, pounds.....	3.28	3.28	0.50	0.50	0.50
Drop test (1-inch screen):					
Held..... per cent.....	84.4				
Passed..... do.....	15.6				
Weather test:					
Time exposed..... days.....	201	201	201	201	201
Condition.....	B	B	B	B	B

Chemical analyses of coal and briquets.

	Test 130.	Test 130.
Laboratory No.....		
Proximate:		
Moisture..... per cent.....	2.62	1.99
Volatile matter..... do.....	37.09	36.56
Fixed carbon..... do.....	54.32	54.14
Ash..... do.....	5.97	7.30
Sulphur..... do.....	1.17	1.12
Ultimate:		
Hydrogen..... do.....	5.08	4.84
Carbon..... do.....	79.02	77.39
Nitrogen..... do.....	1.70	1.53
Oxygen..... do.....	6.87	7.64

Extraction analyses.

	Pitch.	Fuel.	Briquets.
		Tenn. No. 1.	Test 130.
Laboratory No.	3410		
Air-drying loss. per cent.		3.50	1.00
Extract by CS_2 (as received) do.	79.96	2.15	4.54
Pitch in briquet by analysis do.			3.07
Heat value. B. t. u.	16,478	12,869	13,728

TENNESSEE NO. 4.

Bituminous run-of-mine coal from a mine situated 3 miles north of Oliver Springs, Roane County, on the Louisville & Nashville Railroad, was designated Tennessee No. 4, and a part of the sample was used in briquetting test 120; part was mixed with miscellaneous No. 5 (coke breeze) in briquetting tests 150 and 151. Steaming test 405 was made of briquets from briquetting test 120.

In test 120, briquets made on the English machine at a temperature of 179.6° F., with an average weight of 3.3 pounds and with 5 per cent of binder, were easily broken, showing cracks with soft edges; briquets made with 5.5 per cent of binder showed improvement, but the edges were not sharp, and the briquets were easily broken; those with 6 per cent of binder were satisfactory, showing firm edges, hard surfaces, and clean, sharp fracture; those with 6.5 per cent of binder showed improvement in surfaces and edges. In the drop test with briquets containing 5 per cent of binder the 1-inch screen held 83.3 per cent; with 5.5 per cent of binder 85.4 per cent was held. Briquets made on the Renfrow machine, at a temperature of 149° F., an average weight 0.4 pound, and 6, 6.5, and 7 per cent of binder, had smooth, firm surfaces and slightly soft, though generally satisfactory, edges; 7.5 per cent of binder made the best briquet of the lot; all surfaces and fractures were entirely satisfactory.

In test 150, 80 per cent of Tennessee No. 4 and 20 per cent of miscellaneous No. 5 (coke breeze) were used. Briquets with 8 per cent of binder were satisfactory, with no difference in appearance from those made from Tennessee No. 4 alone. They were tough when warm, easily handled and stored, and had rough but hard surfaces. The briquets were difficult to break, and the fracture was uneven, although the edges were firm. The structure was porous.

In test 151, 75 per cent of Tennessee No. 4 and 25 per cent of miscellaneous No. 5 were used. With the increased percentage of coke breeze the briquets were easily broken when warm, although they were sufficiently tough after cooling.

Briquetting tests.

	Test No.		
	120.	150.	151.
Size as used:			
Over $\frac{1}{4}$ inch.....per cent..	3.2	3.2	3.2
$\frac{1}{4}$ inch to $\frac{1}{2}$ inch.....do..	11.4	10.9	10.9
$\frac{1}{2}$ inch to $\frac{3}{4}$ inch.....do..	19.9	18.6	18.6
$\frac{3}{4}$ inch to 1 inch.....do..	25.0	25.8	25.8
Under $\frac{1}{4}$ inch.....do..	40.0	41.5	41.5
Details of manufacture:			
Machine used.....	(a)	Renfrow No. 1.	Renfrow No. 1.
Temperature of briquets.....°F..	(a)	149	149
Binder—			
Kind.....	W. G. P.	W. G. P.	W. G. P.
Laboratory No.....	3410	3410	3410
Amount.....per cent..	(a)	8	8
Weight of—			
Fuel briquetted.....pounds..	44,000	40,000	40,000
Briquets, average.....do..	(a)	.532	.513
Weathering test:			
Time exposed.....days..	214	214	214
Condition.....	B	B	B

(a) See notes preceding table.

Chemical analyses of coal and briquets.

	Tennes- see No. 4.	Miscel- laneous No. 4.	Test 120.	Test 150.	Test 151.
Laboratory No.....	3058	3386	-----	3390	3367
Proximate:					
Moisture.....per cent..	6.39	10.72	2.88	2.55	2.71
Volatile matter.....do..	32.32	4.08	35.78	31.91	29.56
Fixed carbon.....do..	51.76	73.25	53.84	57.42	53.41
Ash.....do..	9.53	11.95	7.50	8.12	9.32
Sulphur.....do..	.98	1.02	.91	.92	.93
Ultimate:					
Hydrogen.....do..	5.02	-----	4.73	4.34	4.15
Carbon.....do..	74.95	-----	77.93	78.53	78.40
Nitrogen.....do..	1.67	-----	1.79	1.71	1.67
Oxygen.....do..	7.13	-----	6.89	6.15	5.90

Extraction analyses.

	Pitch.	Fuel.		Briquets.		
		Tennes- see No. 4.	Miscel- laneous No. 5.	Test 120.	Test 150.	Test 151.
Laboratory No.....	3410	3058	3386	-----	3390	3367
Air-drying loss.....per cent..	-----	4.70	8.80	1.10	0.60	0.60
Extracted by CS ₂ (as received).....do..	79.98	1.38	.59	5.75	6.23	6.52
Pitch in briquet by analysis.....do..	-----	-----	-----	5.56	7.29	7.51
Heat value.....B. t. u..	16,478	12,578	11,036	13,578	13,536	13,154

TENNESSEE NO. 7B.

Bituminous slack coal, through a plate perforated with one-fourth by 1 inch holes, from Wilder, Fentress County, on the Southern Railway, was designated Tennessee No. 7B, and 7,000 pounds of the washed sample was used in briquetting test 121. Steaming test 406 was made of the briquets from briquetting test 121.

Briquets with 5.5 and 6 per cent of binder had firm edges, but their surfaces were slightly crumbly when rubbed; they broke with a clean fracture, leaving a firm surface; those with 6.5 and 7 per cent of binder had hard and smooth surfaces, with a hard and firm fracture. Not enough coal was furnished to make briquetting tests on the Renfrow No. 1 machine or steaming tests of unbriquetted fuel.

Briquetting tests.

	Test No.			
	121A.	121B.	121C.	121D.
Size as used:				
Over $\frac{1}{2}$ inch.....per cent.	1.8			
$\frac{1}{2}$ inch to $\frac{3}{4}$ inch.....do.	10.8			
$\frac{3}{4}$ inch to $\frac{1}{2}$ inch.....do.	19.0			
$\frac{1}{2}$ inch to $\frac{1}{4}$ inch.....do.	21.0			
Through $\frac{1}{4}$ inch.....do.	47.4			
Details of manufacture:				
Machine used.....	English.	English.	English.	English.
Briquetting temperature.....°F.	179.6	179.6	179.6	179.6
Binder—				
Kind.....	W. G. P.	W. G. P.	W. G. P.	W. G. P.
Laboratory No.....	3410	3410	3410	3410
Amount.....per cent.	5.5	6	6.5	7.0
Average weight of briquet.....pounds.	3.08	3.08	3.08	3.08
Drop test (1-inch screen):				
Held.....per cent.	78.3	79.9	80.5	95.2
Passed.....do.	21.7	20.1	10.5	4.8
Weather test:				
Time exposed.....days.	200	200	200	200
Condition.....	B	B	B	B

Chemical analyses of coal and briquets.

	Car sample.	Test 121.
Laboratory No.....	3186	
Proximate:		
Moisture.....per cent.	7.51	3.37
Volatile matter.....do.	31.74	35.80
Fixed carbon.....do.	50.63	50.77
Ash.....do.	10.12	10.06
Sulphur.....do.	2.26	2.09
Ultimate:		
Hydrogen.....do.		5.02
Carbon.....do.		75.58
Nitrogen.....do.		1.36
Oxygen.....do.		5.47

Extraction analyses.

	Pitch.	Fuel.	Briquets.
		Tenn. No. 7B.	Test 121.
Laboratory No.....	3410	3186	
Air-drying loss.....per cent.		6.10	1.80
Extracted by CS ₂ (as received).....do.	79.98	1.84	6.06
Pitch in briquet by analysis.....do.			5.39
Heat value.....B. t. u.	16,478	12,447	13,090

Chemical analyses of coal and briquets.

	Car sam- ple 9B.	Car sam- ple 9C.	Test 122.
Laboratory No.	3114	3115	3338
Proximate:			
Moisture..... per cent..	5.68	4.68	4.24
Volatile matter..... do..	25.36	28.75	29.80
Fixed carbon..... do..	50.41	57.31	56.10
Ash..... do..	18.55	9.26	9.86
Sulphur..... do..	0.74	0.65	0.79
Ultimate:			
Hydrogen..... do..	4.65	4.74	4.60
Carbon..... do..	64.87	77.48	77.00
Nitrogen..... do..	1.27	1.30	1.43
Oxygen..... do..	5.16	6.09	5.84

Extraction analyses.

	Pitch.	Fuel.		Briquets.
		Tenn. No. 9B.	Tenn. No. 9C.	Test 122.
Laboratory No.	3410	3114	3115	3338
Air-drying loss..... per cent..		4.70	3.30	2.50
Extracted by CS ₂ (as received)..... do..	79.98	0.19		5.31
Pitch in briquet by analysis..... do..				6.41
Heat value..... B. t. u..	16,478	11,460	13,168	13,047

TENNESSEE NOS. 9B AND 9C (WASHED SLACK).

Bituminous slack coal from Coalmont, Grundy County, on the Nashville, Chattanooga & St. Louis Railway, was designated Tennessee No. 9B after washing, and 14 tons of slack as shipped was designated Tennessee No. 9C. A mixture of 9B and 9C was used in briquetting test 122. Steaming test 393 was made of briquets from the above test.

In test 122 the size of the coal used was as follows: Over one-fourth inch, 1.2 per cent; one-tenth inch to one-fourth inch, 5.5 per cent; one-twentieth to one-tenth inch, 12.5 per cent; one-fortieth inch to one-twentieth inch, 23.8 per cent; through one-fortieth inch, 57 per cent. The English machine was used, the temperature of the briquets being 179.6° F.; kind of binder, water-gas pitch; laboratory number, 3410; weight of fuel briquetted, 14,400 pounds; average weight of briquets, 3.24 pounds.

With 5.4 and 5.6 per cent of binder, the briquets were fairly good, but somewhat crumbly at the edges. With 6.6, 7, and 8 per cent of binder, the outer surfaces were very firm and smooth, the fracture was coarse but not crumbly, and the broken surfaces were very hard. In the drop test, with 7 per cent of binder, the 1-inch screen held 85.6 per cent. In the weathering test the condition of all binders after being exposed 222 days was B. Not enough coal was furnished to make briquets on the Renfrow No. 1 machine for comparative tests.

TENNESSEE NO. 10.

Bituminous slack coal from a mine located 1 mile north of Orme, Marion County, on the Nashville, Chattanooga & St. Louis Railway, was designated Tennessee No. 10. A part of the sample was used in briquetting test 128. Steaming tests 407 and 408 were made of the briquets from the above test.

In test 128 the details were as follows: Condition of coal, 1-inch slack, washed; kind of binder, water-gas pitch; laboratory number, 3410; weight of fuel briquetted, 35,400 pounds. The briquets made with the three percentages of pitch (5.5, 6.5, and 6.7) were all good.

Briquets made on the English machine at a temperature of 175° F. and with 5.5 per cent of binder had an average weight of 3.63 pounds and could be roughened slightly by rubbing; their outer surfaces crumbled slightly when broken. Briquets with 6 and 6.5 per cent of binder could be roughened slightly by rubbing; their outer surfaces were hard and smooth, and their broken surfaces very hard and rough. In the drop test of briquets with 5.5 per cent of binder the 1-inch screen held 81.7 per cent, with 6 per cent of binder the screen held 90.1 per cent, and with 6.5 per cent of binder the screen held 89.1 per cent. In the weathering test all briquets were exposed 203 days, when their condition was B.

Renfrow No. 1 machine briquets made at 149° F. with 6 per cent of binder had an average weight of 0.53 pound and showed crumbly, fractured surfaces. With 6.5 per cent of binder the briquets were tougher and did not break as easily, and broken surfaces were firm, but the briquets seemed to lack pitch. In the weathering test briquets with both binders were exposed 204 days, when their condition was B.

Chemical analyses of coal and briquets.

	Fuel.	Test 128.	Test 128.
Laboratory No.	3359		
Proximate:			
Moisture..... per cent.	7.02	2.46	2.88
Volatile matter..... do.		30.89	30.84
Fixed carbon..... do.		51.80	49.55
Ash..... do.	13.75	14.85	16.73
Sulphur..... do.	.98	1.02	.96
Ultimate:			
Hydrogen..... do.		4.47	4.56
Carbon..... do.		71.53	69.82
Nitrogen..... do.		1.30	.92
Oxygen..... do.		6.43	6.48

Extraction analyses.

	Pitch.	Fuel, Tenn. No. 10.	Briquets, test 128.
Laboratory No.	3410	3359	
Air-drying loss..... per cent.		5.20	1.3
Extracted by CS ₂ (as received)..... do.	79.98	1.32	6.98
Pitch in briquet by analysis..... do.			7.19
Heat value..... B. t. u.	16,478	11,596	11,920

TEXAS NO. 4.

Lignite run-of-mine coal from Hoyt, Wood County, on the Missouri, Kansas & Texas Railway, was designated Texas No. 4. A part of the sample was used in briquetting test 112 at St. Louis. The briquets were too unsatisfactory for use in steaming tests. No analysis was made of the briquets.

The details of test 112 were as follows: Machine used, English; temperature of briquets, 179.6° F.; kind of binder, coal-tar pitch; laboratory number, 2933; percentage of binder, 5, 6, 7, and 8 per cent; weight of fuel briquetted, 4,500 pounds.

The briquets were of a brown color and full of cracks. The different percentages of binder had no relative effect on their firmness; all fell apart when the pitch hardened. None burned well or was satisfactory. They did not swell nor crack, but gradually disintegrated in the fire, burning slowly with characteristic lignite sparking. In the weathering test all briquets were exposed 75 days, when their condition was D.

The unsatisfactory results obtained with this fuel were probably due to the excessive proportion of moisture present. With the English machine it was impossible to control the moisture added in the mixing and softening of the pitch, and as no drying was done before briquetting, the affinity of lignite for moisture caused a large absorption of moisture; as the briquets cooled they also dried to some extent, resulting in the cracking of the briquets mentioned above. With the lignite dried sufficiently before briquetting and with a suitable mixer and press, much more satisfactory briquets could probably have been made from this fuel, as the results obtained were in large measure due to the unsuitability of the briquetting equipment for use with lignite.

Chemical analysis.

	Car sample.		Car sample.
Laboratory No.....	2717	Ultimate:	
Proximate:		Hydrogen.....per cent..	4.41
Moisture.....per cent..	33.85	Carbon.....do.....	65.18
Volatile matter.....do....	27.50	Nitrogen.....do.....	1.07
Fixed carbon.....do.....	31.35	Oxygen.....do.....	17.53
Ash.....do.....	7.30		
Sulphur.....do.....	.51		

Extraction analyses.

	Pitch.	Fuel, Tex. No. 4.
Laboratory No.....	2933	2717
Air-drying loss.....per cent..		26.70
Extracted by CS ₂ (as received).....do....	64.20	
Heat value (as received).....B. t. u..	15,937	7,497

TEXAS—PITTSBURGH NO. 7.

A 3-car shipment of lignite from the Big Lump mine, $3\frac{1}{2}$ miles northeast of Rockdale, Milam County, Tex., was designated Pittsburgh No. 7.

House-heating boiler tests H23, H24, H30, H45, and H46 were made of this fuel, but the larger part of the shipment was used in preliminary briquetting experiments to adjust the drier, cooler, and press of the German plant. This lot was the first to arrive at the Pittsburgh testing station. Considerable difficulty was experienced in making it into briquets, and the shipment was exhausted before satisfactory results were obtained. It is hoped that further tests of this lignite may be made under more favorable conditions.

Mr. Lehmann, the representative of the German company that furnished the lignite-briquetting plant, remained at Pittsburgh until the plant was in good working condition. The tests of the first sample of lignite were therefore under his direction. Most of these tests were preliminary, and it was impossible to obtain a complete record of them, so only a general statement of conditions and results can be given.

The briquetting material in the various tests contained different proportions of water, ranging from 2.3 to 11.2 per cent, and different pressures, from very light to the heaviest possible, were applied to it. From some of the tests the briquets were poorly formed, rough, and scaly, but were fairly strong; from other tests the briquets were well formed but weak.

Only one cohesion (drop) test was made of the briquets obtained, because most of the tests gave only pieces of briquets. In the one test made only 24 per cent was held by a 1-inch screen.

The best briquets were made by using material that contained 11.2 per cent of moisture and applying a rather light pressure. The results seemed to show that for the best results with the German machine the ground and dried lignite should contain between 8 and 12 per cent of moisture, and that the pressure used should be moderate—about that obtained by using a set of dies so ground that in a length of about 80 mm. the decrease in height would be 3 to 4 mm.

As a general conclusion, it may be said that this lignite is deficient in natural binding material, and although it can be briquetted without artificial binder on a machine of the type used, such a machine did not make briquets of satisfactory form and strength from it. To determine the best conditions for briquetting, further tests are needed. It is probable that the lignite will make entirely satisfactory briquets if an artificial binder and a different type of press are used.

The following proximate analyses show the changes in composition of the sample during transportation from the mine to the plant, a

period of 29 days, and after storage at the plant for 17 and 38 days, respectively:

Chemical analyses.

	Mine sample.	Car sample.	Plant samples.	
			After 17 days.	After 38 days.
Date taken.....	Feb. 1	Mar. 1	Mar. 18	Apr. 8
Laboratory No.....	7270	7350	7433	7554
Proximate:				
Air-drying loss..... per cent.	31.50	25.10	25.70	25.30
Moisture..... do.	35.30	33.38	31.72	30.98
Volatile matter..... do.	26.22	27.44	26.81	29.04
Fixed carbon..... do.	29.58	29.62	31.97	30.41
Ash..... do.	8.90	9.56	9.50	9.57
Sulphur..... do.	.76	.94	.91	.92
Ultimate:				
Hydrogen..... do.		4.59		
Carbon..... do.		62.01		
Nitrogen..... do.		1.01		
Oxygen..... do.		16.63		
Extract by CS ₂ (moisture free)..... do.		1.23		1.16
Heat value..... B. t. u.	6,898	7,189	7,423	7,301

TEXAS—PITTSBURGH NO. 8.

A sample shipment, 2 cars, of $\frac{1}{2}$ -inch lump lignite from the Carr mine No. 3, 2 miles southwest of Lytle, Medina County, Tex., was designated Pittsburgh No. 8 and was tested on the German press without binder.

This lignite had a dark-brown color. It weathered very rapidly, exposure to a single rainstorm causing lumps to crack in all directions. After it had been wet and dried a few times the sample crumbled entirely to slack.

Gas-producer test 180 and briquetting tests 293, 295, and 296 were made of this fuel, and several preliminary briquetting tests for adjusting the German press when being installed were also made.

Several preliminary tests were made under the direction of Mr. Lehmann. In the first of these, with a steam pressure of 5 pounds in the drier, the percentage of moisture in the fuel was reduced from 30 per cent to 11.2 per cent. A rather low pressure in the molds produced well-formed but weak briquets.

Material containing 10 per cent of moisture, and subjected to a high pressure, furnished briquets that had poor form and rough surfaces, but they were much stronger than those made in the first test. The roughness of surface was largely caused by steam (generated while the briquet was being formed under high pressure) that blew out around the sides of each briquet just before it left the mold. The dust blown out by the steam was disagreeable to breathe and at times formed clouds that darkened the whole room.

In the next test, material containing 8.4 per cent of moisture was subjected to a high pressure. The briquets were better formed and

smoother than in the previous test, and were the strongest yet made from the lignite.

In the fourth test the material contained 6 per cent of moisture, but the same high pressure was used. The briquets were still better, being well formed and strong. Another test, with the material containing only 3.6 per cent of moisture, gave the best briquets obtained from this fuel. However, with this low-moisture content the mold pressure was too great for the press, and after an hour's run the main bearing of the connecting rod became so hot that the press had to be stopped.

The following table summarizes the essential data of the tests made with this sample of lignite:

Summary of tests.

Test No.	Moisture in briquet material.	Pressure in mold.	Character of briquets.		Remarks.
			Form.	Strength.	
	<i>Per cent.</i>				
Preliminary.....	11.2	Low.....	Good.....	Weak.....	Pressure insufficient.
Do.....	10.0	High.....	Poor.....	Strong.....	Briquets rough.
Do.....	8.4	do.....	Fair.....	do.....	
Do.....	6.0	do.....	Good.....	do.....	Better than previous lot.
Do.....	3.6	do.....	do.....	do.....	Best lot.
293.....	8.7	do.....	do.....	Good.....	
295.....	7.7	Very high.....	None made.....		Material too dry.
296.....	12.2	do.....	Fair.....	Weak.....	Material too fine.

In the first official test (293) the material contained 8.7 per cent of moisture, and a high pressure was applied. About 61 per cent of the material that reached the press was coarser than one-twentieth inch, but practically all of this 61 per cent passed through a screen with $\frac{1}{4}$ -inch mesh. The results indicated that it is well to have the material as evenly sized as possible and containing little dust. If the proportion of dust is excessive, the material does not bind well and blows out of the mold when pressed. Even sizing is also desirable because it permits even drying. The writer believes that this fuel should be crushed so that it will all pass through a $\frac{1}{4}$ -inch screen, about 60 per cent being retained on a $\frac{1}{16}$ -inch screen.

In tests 295 and 296 the dies were set to give a very high pressure. In test 295 the briquetting material contained 7.7 per cent moisture and was probably too dry; it did not bind, but fell from the press in the form of powder, accompanied by clouds of dust.

In test 296 the dried material had 12.2 per cent of moisture. The material probably contained too much dust, for the briquets crumbled easily and were not strong enough to be handled.

Plates XVI to XXI show how the briquets from tests 293 and 296 withstood exposure to the weather.

This lignite was made into excellent briquets without the addition of any binder. The moisture content of the dried material should be between 8 and 10 per cent if a moderate pressure is used, but only about 5 per cent, or even less, if an extremely high pressure is used. For best results the writer recommends that the material be pulverized so that about 60 per cent of the particles are between one-fourth inch and one-twentieth inch in diameter, that it go to the press containing 10 per cent of moisture, and that the briquetting pressure be 20,000 pounds per square inch.

The best briquets made from this fuel had strength enough to stand handling. They took the shape of the mold well, their edges were firm, and they had jet-black, lustrous surfaces. Such briquets should make a satisfactory steam or household fuel. No steaming tests were made of either the raw or the briquetted lignite. The ash content (about 17 per cent) of the briquets is considerably higher than that of good-quality bituminous coal.

Briquetting this lignite should improve its heating value 30 to 40 per cent by reducing the percentage of moisture. On the assumption that in briquetting the uncombined moisture is reduced from 30 per cent to 10 per cent, 1 ton (2,000 pounds) of raw fuel will make 1,600 pounds of briquets, or 2,500 pounds of raw fuel will make 1 ton of briquets. The raw fuel has a heating value of 6,800 British thermal units per pound and the briquetted fuel has a heating value of 9,300 British thermal units; therefore the fuel value of the briquets is 37 per cent higher than that of the raw fuel. Moreover, the briquets have the following additional advantages over raw fuel:

(a) Being of uniform size, they burn more freely and give off less smoke, a decided merit when used as a household fuel in a residence district.

(b) The briquets resist the effects of the weather much better than the raw fuel, and therefore can be stored for a longer time without serious deterioration. The briquets are not, however, much more waterproof than the raw fuel, and should be stored under cover; there they will remain in perfect condition for several months at least, whereas the raw fuel under similar conditions will disintegrate rapidly.

(c) The cost of transporting the briquetted fuel should be only 80 per cent of the cost of transporting enough raw fuel to furnish the same heating value.

Briquetting tests.

	Test No.		
	293.	295.	299.
Size as shipped.....	$\frac{1}{2}$ " lump.	$\frac{1}{2}$ " lump.	$\frac{1}{2}$ " lump.
Size as used:			
Over $\frac{1}{2}$ inch.....per cent..	0.5	0.5	0.5
$\frac{1}{2}$ inch to $\frac{3}{4}$ inch.....do..	30.5	18.0	18.0
$\frac{3}{4}$ inch to $\frac{1}{2}$ inch.....do..	30.0	28.5	28.5
$\frac{1}{2}$ inch to $\frac{3}{4}$ inch.....do..	24.5	21.0	21.0
Through $\frac{3}{4}$ inch.....do..	14.5	32.0	32.0
Details of manufacture:			
Machine used.....	German.	German.	German.
Briquetting temperature.....°F..	90	94	101
Binder used.....	None.	None.	None.
Tangent of die angle.....	5.5/89	6/89	6/89
Steam pressure on drier.....pounds..	13	15	11
Weight of—			
Fuel briquetted.....do..	11,440	None.	4,840
Briquets, average.....do..	0.9		0.7
Moisture in briquet mixture.....per cent..	8.73	7.68	12.17
Drop test (1-inch screen):			
Held.....do..	44	(a)	(b)
Passed.....do..	66		
Tumbler test (1-inch screen):			
Held.....do..	37		(b)
Passed.....do..	63		
Fines through 10-mesh sieve.....do..	74		
Weather test:			
Time exposed when examined.....days..	66		
Condition when examined.....	D		

a No briquets.

b Briquets too weak for cohesion test.

The changes in composition of the lignite during transportation from the mines to the plant (25 days) and during storage (42 days) at the plant till used for one of the last tests are shown by the following analyses; analyses of the briquets or briquetting mixture are also included:

Chemical analyses of coal and briquets.

	Mine sample.	Car sample.	Sample as used on last test.	Test 293.	Test 293. ^a	Test 296.
Date taken.....	Feb. 5.	Mar. 2.	Apr. 13.			
Laboratory No.....	7330	7461	7584	7583	7585	7623
Proximate:						
Air-drying loss.....per cent..	24.20	26.30	19.40	1.50		
Moisture.....do..	32.92	30.77	28.00	8.07	5.02	9.96
Volatile matter.....do..	27.42	27.36	27.50	37.31	38.53	37.84
Fixed carbon.....do..	27.08	28.39	32.30	35.87	37.81	35.99
Ash.....do..	12.58	13.48	12.20	18.75	18.64	16.21
Sulphur.....do..	1.46	1.62	1.60	1.97	2.18	1.96
Ultimate:						
Hydrogen.....do..		4.20				
Carbon.....do..		57.58				
Nitrogen.....do..		1.04				
Oxygen.....do..		15.37				
Extract by CS ₂ (moisture free).....do..			1.91	1.98		2.14
Heat value.....B. t. u..	6,840	7,079	7,580	9,306		9,223

^a Analysis of the material which flaked off from the surface of the briquets.**TEXAS—PITTSBURGH NO. 9.**

A sample shipment, 3 cars, of $\frac{1}{2}$ -inch lump lignite from the Calvert mine, 6 miles west of Calvert, Robertson County, Tex., was designated Pittsburgh No. 9.

Gas-producer tests 181 and 182, house-heating boiler tests H41 and H43, and briquetting tests 297, 320, 321, 323, 327, and 328 were made with the German lignite press, using the raw lignite without binder.

Test 297 did not yield any briquets. Seemingly the pressure in the mold was excessive; there may have been too little moisture in the material or the die angle may have been too great.

In test 320 a high pressure was used. The briquets were poorly formed and too weak to stand handling. The moisture content of the material was manifestly too high for the pressure used. Much dust was blown out around the sides of each briquet.

In test 321 the same pressure was used, but the material was still too moist and no satisfactory briquets were obtained. The briquets made had poor forms and were too weak to stand handling.

The material was drier in test 323 than in tests 297, 320, and 321, and contained only 3 per cent of moisture when it reached the machine. It was also finer. This material, however, stalled the press. After several attempts to operate the machine the test was stopped and the material remaining in the hopper was thrown away.

In test 327 the material was dried less than in test 323, and contained 4.6 per cent moisture when it reached the machine. The dies were changed so as to give less pressure. The briquets obtained were better than those made in test 323, but although they had good form they lacked strength, and crumbled when handled.

In the next test (328) the percentage of moisture in the briquetting material was raised to 6.4 per cent with beneficial results. The same pressure was used as in test 327. The briquets were the best so far obtained, but they were not strong enough to stand handling. The test was not completed, however, for a strong steel brace having a cross section of 3 by 5½ inches snapped and put the press out of operation until another brace could be forged. The tests were suspended before other pressures could be tried.

Following is a summary of the most important features of the tests of this sample of lignite:

Summary of tests.

Test No.	Moisture in briquet material.	Pressure in mold.	Character of briquets.		Remarks.
			Form.	Strength.	
	<i>Per cent.</i>				
327	4.6	High.....	Good.....	Weak.....	Too much moisture. Better than test 327. Too much pressure. Stalled machine; too little moisture and too much pressure. Too much moisture and dust. Do.
328	6.4	do.....	do.....	do.....	
297	7.0	do.....	None made.....	do.....	
323	3.0	Very high.....	do.....	do.....	
321	6.4	do.....	Poor.....	Weak.....	Too much moisture and dust. Do.
320	9.47	do.....	do.....	do.....	

It will be seen that the moisture content in the briquetting material varied from 3 per cent to 9½ per cent. Two different pressures were used, but, although some of the tests gave briquets of good form, no briquets were of satisfactory strength.

To yield the best results with the German machine, this lignite should be ground so that it will all pass a ¼-inch screen, but about 60 per cent will be retained by a ⅜-inch screen; it should be dried till it contains 10 per cent of moisture. The die angle should be small enough to give a moderate pressure. However, the indications are that this lignite does not contain enough natural binder to make good briquets on the German press, and it should be tested with a little artificial binder on a press similar to the English machine. The writer is inclined to doubt whether this lignite can be briquetted at all without the addition of binding material. However, further tests should be made before a final conclusion is reached.

Briquetting tests.

	Test No.					
	297.	320.	321.	323.	327.	328.
Size as shipped.....	¾" lump.	¾" lump.	¾" lump.	¾" lump.	¾" lump.	¾" lump.
Size as used:						
Over ½ inch..... per cent.	2.0	3.0	2.5	2.5	2.0	2.0
⅜ inch to ½ inch..... do.	18.0	23.0	25.0	30.0	31.5	30.5
⅜ inch to ⅜ inch..... do.	30.0	28.0	28.5	28.5	28.0	30.5
⅜ inch to ⅜ inch..... do.	27.0	22.5	21.0	20.5	19.5	19.0
Through ⅜ inch..... do.	23.0	23.5	23.0	18.5	18.0	18.0
Details of manufacture:						
Machine used.....	German.	German.	German.	German.	German.	German.
Briquetting temperature..... °F.	84	100	102	97	93	116
Binder used.....	None.	None.	None.	None.	None.	None.
Tangent of die angle.....	6/89	7/89	7/89	7/89	6/89	6/89
Steam pressure on drier..... pounds.	10	4	10	15	10	12
Weight of—						
Fuel briquetted..... do.	None.	None.	None.	None.	None.	None.
Briquets, average..... do.						
Moisture in briquet mixture.... per cent.	6.98	9.47	6.40	3.0	4.6	6.4

The following proximate analyses show the changes in composition of this lignite during transportation from the mine to the plant, a period of 26 days, and in storage at the plant for 81 days until used for the last test. The fourth column gives an average of six analyses made on different dates.

Chemical analyses of coal and briquets.

	Mine sample.	Car sample.	Sample as used on last test.	Average of 6 samples.	Briquet 320.	Briquet 321.
Date taken.....	Mar. 6	Apr. 1	June 21			
Laboratory No.....	7403	7513	7950		7904	7941
Proximate:						
Air-drying loss.....per cent..	29.30	25.40	12.10	13.17		
Moisture.....do.....	34.33	29.62	23.50	24.01	9.04	7.92
Volatile matter.....do.....	25.94	27.60	29.07	29.75	37.22	37.33
Fixed carbon.....do.....	30.93	29.37	35.74	34.74	40.86	41.75
Ash.....do.....	8.80	13.41	11.69	11.50	12.88	13.00
Sulphur.....do.....	.95	.98	.82	.91	1.15	1.16
Ultimate:						
Hydrogen.....do.....		4.45				
Carbon.....do.....		57.21				
Nitrogen.....do.....		.98				
Oxygen.....do.....		16.92				
Extract by CS ₂ (moisture free).....do.....				1.28		1.14
Heat value.....B. t. u.....	7,214	7,040	8,089	8,021	9,574	9,929

UTAH NO. 1.

Bituminous run-of-mine coal from a prospect on Huntington Creek, Carbon County, was designated Utah No. 1. A part of the sample as received at the plant was used in briquetting test 126; also in a mixture with Rhode Island No. 1 in briquetting test 127. Steaming tests 414 and 415 were made of briquets from test 127.

In test 126 the size of coal used was as follows: Over one-fourth inch, 1.4 per cent; one-tenth inch to one-fourth inch, 5.6 per cent; one-twentieth inch to one-tenth inch, 14.4 per cent; one-fortieth inch to one-twentieth inch, 25 per cent; through one-fortieth inch, 53 per cent. This coal was briquetted with low percentages of binder, with the idea of utilizing the natural resin (probably copalite) contained in the coal. With 4 and 5 per cent of binder the heat in the machine would not melt the natural resin, and the briquets were very deficient in binder and crumbled badly. With 6 per cent of binder the surfaces of whole briquets and of fractures crumbled. The use of a still higher percentage of pitch would improve the briquets. No drop tests were made of these briquets and no other regular data were obtained, on account of the small size of the sample.

In test 127 (with Rhode Island No. 1) the size of coal used was as follows: Over one-fourth inch, 1 per cent; one-tenth inch to one-fourth inch, 5.8 per cent; one-twentieth inch to one-tenth inch, 9.4 per cent; one-fortieth inch to one-twentieth inch, 26.4 per cent; through one-fortieth inch, 57.4 per cent. This test was made to prove the value of briquetting a good fuel with one that is commercially worthless. A high volatile coal low in ash was chosen to mix with the graphitic coal. Various percentages were tried, but 47 per cent of each coal and 6 per cent of binder made an entirely satisfactory briquet. Six

per cent of binder made excellent briquets, whose outer surface was smooth, lustrous, and very hard. The briquets broke without crumbling, and the broken surfaces were smooth and hard.

Briquetting tests.

	Test 126.	Test 127.
Details of manufacture:		
Machine used.....	Renfrow No. 1.	Renfrow No. 1.
Temperature of briquets.....°F.	149	149
Binder—		
Kind.....	W. G. P.	W. G. P.
Laboratory No.....	3410	3410
Amount.....per cent.	4, 5, 6	6
Weight of—		
Fuel briquetted.....pounds..	2,000	16,000
Briquets, average.....	0.43	0.5
Weathering test:		
Time exposed.....days..	214	214
Condition.....	C.	B.

Chemical analyses of coal and briquets.

	Utah No. 1.	Rhode Island No. 1.	Test 127.	Test 127.
Laboratory No.....	3199	3216		
Proximate:				
Air dry loss.....per cent..	3.80	2.0	0.8	0.8
Moisture.....do.....	6.05	2.41	2.45	2.27
Volatile matter.....do.....	42.02	4.92	24.21	22.20
Fixed carbon.....do.....	47.06	73.61	62.60	65.29
Ash.....do.....	4.87	19.06	10.74	10.24
Sulphur.....do.....	.55	.07	.41	.41
Ultimate:				
Hydrogen.....do.....	5.42	.65	2.95	3.13
Carbon.....do.....	76.98	76.95	78.63	77.64
Nitrogen.....do.....	1.47	.17	.74	.74
Oxygen.....do.....	10.36	2.63	6.25	7.59
Extract by CS ₂ (as received).....do.....	3.83	.02	6.89	6.89
Pitch in briquets by analysis.....do.....			6.40	6.40
Heat value.....B. t. u..	13, 151	10, 996	12, 532	12, 532

UTAH NO. 2.

Subbituminous slack (through 1½-inch screen) from Coalville, Summit County, on the Union Pacific Railroad, was designated Utah No. 2. A part of the sample as received at the plant was used in briquetting test 132; part was mixed with Rhode Island No. 1 in briquetting test 133. Steaming tests 402 and 404 were made of briquets from briquetting test 132 and steaming test 416 of briquets from briquetting test 133.

In test 132 this coal, sent primarily for briquetting, had all the characteristics of lignite. Oil shale was sent to be used as a binder, but all efforts to use it for this purpose utterly failed. The briquets made on the English machine with 5 and 6 per cent of binder indicated a shortage of pitch; their outer surfaces were rough but crumbly and porous; they were crumbly at fractures, and broken and fractured surfaces were not firm. Briquets with 7 per cent of binder

showed improvement over the others but were not satisfactory; their outer surface was harder but could be roughened by rubbing; they did not break clean. In the drop test, with 6 per cent of binder, the 1-inch screen held 54.4 per cent. The Renfrow briquets, with 6 per cent of binder, were soft and showed shortage of pitch. Those with 7 per cent of binder showed improvement. However, they broke with crumbly fracture and their broken surfaces were not firm. Eight per cent of binder made good briquets, with hard outer surface; they broke clean and their broken surfaces were firm.

In test 133 Rhode Island No. 1, the only available high volatile coal, was chosen for a test with a view to supplementing the data obtained in test 127. The test was not successful, as the coal showed characteristics of lignite, both in briquetting and in burning. The mixture contained 47 per cent of each coal. Briquets with 6 per cent of binder were tough and hard; their outer surface was smooth and very hard and their fracture rough but clean and firm. No drop tests were made.

Briquetting tests.

	Test 132.	Test 132.	Test 133.
Size as used:			
Over $\frac{1}{2}$ inch.....per cent..	0.8		1.2
$\frac{1}{2}$ inch to $\frac{3}{4}$ inch.....do..	8.0		6.0
$\frac{3}{4}$ inch to 1 inch.....do..	21.4		19.2
1 inch to 1 $\frac{1}{4}$ inch.....do..	29.8		25.6
Through 1 $\frac{1}{4}$ inch.....do..	40.0		48.0
Details of manufacture:			
Machine used.....	English.	Renfrow No. 1.	Renfrow No. 1.
Briquetting temperature.....°F..	175	149	149
Binder—			
Kind.....	W. G. P.	W. G. P.	W. G. P.
Laboratory No.....	3410	3410	3410
Amount.....per cent..	6, 7, 8	5, 6, 7	6
Weight of—			
Fuel briquetted.....pounds..	29,000	29,000	37,000
Briquet, average.....do..	3.03	0.42	0.52
Weather test:			
Time exposed.....days..	190	190	190
Condition.....	D	C	E

Chemical analyses of coals and briquets.

	Utah No. 2.	Rhode Island No. 1.	Test 132. ^a	Test 132. ^b	Test 133.
Laboratory No.....	3259	3216			
Proximate:					
Moisture.....per cent..	12.66	2.41	11.53	9.32	5.85
Volatile matter.....do..	38.30	4.92	38.91	39.68	25.20
Fixed carbon.....do..	43.19	73.61	43.83	45.06	59.56
Ash.....do..	5.85	19.06	5.73	5.95	9.39
Sulphur.....do..	1.39	.07	1.36	1.37	.85
Ultimate:					
Hydrogen.....do..		.65	4.70	4.75	2.94
Carbon.....do..		76.95	72.15	73.30	76.42
Nitrogen.....do..		.17	1.27	1.27	.73
Oxygen.....do..		2.63	13.86	12.61	9.04

^a Analysis of English briquets.

^b Analysis of Renfrow briquets.

Extraction analyses.

	Pitch.	Fuel.		Briquets.	
		Utah No. 2.	Rhode Island No. 1.	Test 132.	Test 133.
Laboratory No.....	3410	3259	3216		
Air-drying loss..... per cent.		2.30	2.00	1.60	1.40
Extracted by CS ₂ do.	79.98	.26	.02	4.61	6.06
Pitch in briquet by analysis..... do.				5.46	7.50
Heat value..... B. t. u.	16,478	10,697	10,966	11,180	11,527

VIRGINIA NO. 5B.

Semianthracite slack coal from the bed known as the "Big Seam" (Bush Mountain Field), 10 miles west of Blacksburg, Montgomery County, was designated Virginia No. 5B and the sample was used in briquetting tests 187 and 188. Steaming test 494 was made of equal weights of briquets from briquetting tests 187 and 188.

In tests 187 and 188 the size of coal used was as follows: Over one-fourth inch, 0.6 per cent; one-tenth inch to one-fourth inch, 8.8 per cent; one-twentieth inch to one-tenth inch, 23.6 per cent; one-fortieth to one-twentieth inch, 27.6 per cent; through one-fortieth inch, 39.4 per cent. A very hard, nonporous briquet was made with 6 per cent of binder on the English machine. Renfrow briquets with 7 per cent of binder stuck together when piled, showing excess of pitch; otherwise these briquets were satisfactory. This is an exceptionally good briquetting coal, requiring only a small percentage of binder and making briquets with firm, smooth outer surfaces, sharp edges, and characteristic glossy fracture.

Briquetting tests.

	Test 187.	Test 188.
Details of manufacture:		
Machine used.....	Renfrow No. 1.	English.
Briquetting temperature..... °F.	185	185
Binder—		
Kind.....	W. G. P.	W. G. P.
Laboratory No.....	4543	4543
Amount..... per cent.	7	6.25
Weight of—		
Fuel briquetted..... pounds.	8,000	4,000
Briquet, average..... do.	0.457	4.11
Drop test (1-inch screen):		
Held..... per cent.	78.5	85.7
Passed..... do.	21.5	14.3
Tumbler test (1-inch screen):		
Held..... do.	94.0	89.3
Passed..... do.	6.0	10.7
Fines through 10-mesh sieve..... do.	96.9	79.1
Weather test:		
Time exposed..... days.	10	6
Condition.....	A	A
Absorption test:		
Time immersed..... days.	19	19
Water absorbed..... per cent.	14.8	12.6
Average for first 5 days..... do.	2.0	1.12
Specific gravity (apparent).....	1.182	1.209

Chemical analyses of coal and briquets.

	Car sample.	Test 187.	Test 188.
Laboratory No.	4294	4545	4417
Proximate:			
Moisture.....per cent..	7.52	4.52	5.02
Volatile matter.....do.	10.29	14.28	15.52
Fixed carbon.....do.	65.96	65.93	64.51
Ash.....do.	16.23	15.27	14.96
Sulphur.....do.	.65	.79	.84
Ultimate:			
Hydrogen.....do.	8.82	3.68	3.66
Carbon.....do.	74.66	76.17	75.82
Nitrogen.....do.	.75	.81	.88
Oxygen.....do.	2.52	2.51	3.02

Extraction analyses.

	Pitch.	Fuel.	Briquets.	
		Va. 5B.	Test 187.	Test 188.
Laboratory No.	4543	4294	4545	4417
Air-drying loss.....per cent..		7.10	3.60	4.10
Extracted by C ₈do.	99.66	.27	6.24	5.74
Pitch in briquet by analysis.....do.			6.01	5.51
Heat value.....B. t. u..	16,909	11,898	12,542	12,468

VIRGINIA—JAMESTOWN NO. 2

The sample designated as Jamestown No. 2 consisted of several cars of semibituminous run-of-mine coal from a mine working the Pocohontas No. 3 bed, Pocohontas, Tazewell County, Va., on the Norfolk & Western Railway.

Excellent briquets were made from this coal. Those made with 6 per cent of water-gas pitch binder (Lab. No. 5458) stood the cohesion tests as well as those made with 8 per cent of binder (Lab. No. 5563). The warm briquets from all four of the tests had smooth, firm surfaces, and those from tests 251 and 252 fractured with difficulty. Those from tests 254 and 255 were more brittle, though not easily broken. The surface of the briquets from tests 251, 252, and 254 was smooth and firm, whereas the surface of briquets from test 255 crumbled easily. The cold briquets of tests 251, 252, and 254 were excellent, had smooth, firm surfaces, broke without crumbling, and the fractured surfaces were smooth and firm. The cold briquets of test 255 were rather brittle and fractured easily, and the surface of fractures crumbled easily.

The entire shipment of coal was made into briquets and delivered to the U. S. S. *Connecticut* for trial test under marine boilers. The test on the *Connecticut* was conducted in December, 1907, on a run from New York Harbor to Hampton Roads, Va.

Briquetting tests.

	Test 251.	Test 252.	Test 254.	Test 255.
Size as used:				
Over $\frac{1}{2}$ inch.....per cent..	1.3	6.1	0.0	0.0
$\frac{1}{2}$ inch to $\frac{3}{4}$ inch.....do..	2.4	6.3	9.4	7.4
$\frac{3}{4}$ inch to 1 inch.....do..	13.7	22.6	24.2	23.5
1 inch to $\frac{5}{8}$ inch.....do..	27.2	32.3	28.0	28.3
Through $\frac{5}{8}$ inch.....do..	54.4	38.7	38.4	40.8
Details of manufacture:				
Machine used.....	English.	Renfrow No. 2.	Renfrow No. 2.	Renfrow No. 2.
Briquetting temperature.....°F.	192	200	204	205
Binder—				
Kind.....	W. G. P.	W. G. P.	W. G. P.	W. G. P.
Laboratory No.....	5458	5458	5563	5563
Amount.....per cent..	6	6	8	6
Weight of—				
Fuel briquetted.....pounds..	138,101	181,000	2,000	2,000
Briquet, average.....do..	3.27	0.653	0.581	0.596
Moisture in briquet mixture.....per cent..	7.62	3.99		
Drop test (1-inch screen):				
Held.....do..	74.0	66.5	65.0	46.0
Passed.....do..	26.0	33.5	35.0	54.0
Tumbler test (1-inch screen):				
Held.....do..	79.0	69.0	71.0	60.5
Passed.....do..	21.0	31.0	29.0	39.5
Fines through 10-mesh sieve.....do..	93.4	95.5	95.2	95.5
Weather test:				
Time exposed.....days..	171	178	161	161
Condition.....	A	A	A	B
Absorption test:				
Time immersed.....days..	30	21	21	21
Water absorbed.....per cent..	15.38	14.02	15.46	16.62
Average for first four days.....do..	2.04	2.74	2.75	3.29
Specific gravity (apparent).....	1.097	1.108	1.101	1.096

Chemical analyses of coal and briquets.

	Raw fuel.	Test 251.	Test 252.	Test 254.	Test 255.
Laboratory No.....	5333	5466	5454	5455	5457
Proximate:					
Moisture.....per cent..	17.68	13.52	19.73	19.70	19.68
Fixed carbon.....do..	74.19	72.79	72.65	73.08	73.22
Ash.....do..	6.13	6.24	5.96	5.62	5.50
Sulphur.....do..	.59	.63	.61	.50	.58
Ultimate:					
Hydrogen.....do..	4.47	4.37	4.51	4.65	4.22
Carbon.....do..	84.54	82.66	84.86	85.93	84.81
Nitrogen.....do..	.89	.88	.88	.99	.90
Oxygen.....do..	3.24	5.06	3.05	2.21	3.90

Extraction analyses.

	Pitches.		Fuel, J-2.	Briquets.			
				Test 251.	Test 252.	Test 254.	Test 255.
Laboratory No.....	5458	5563	5333	5466	5454	5455	5457
Air-drying loss.....per cent..			1.20	1.80	1.10	1.10	0.90
Extracted by CS ₂ (as received).....do..	92.92	90.33	0.56	5.53	7.75	7.65	5.87
Pitch in briquet by analysis.....do..				5.71	5.62	7.90	5.82
Heat value.....B. t. u..	16,993	16,718	14,682	14,549	14,726	14,794	14,796

VIRGINIA—JAMESTOWN NO. 14.

The sample designated as Jamestown No. 14 consisted of a car of Virginia semianthracite culm from a mine working the Big Vein, Merrimac, Montgomery County, Va., on the Virginia Anthracite Railway.

The sample was tested at the Norfolk plant.

As received, the sample was very wet, and serious difficulty was experienced in crushing it with the Williams mill because of its tendency to pack and stop the mill. About 11 tons of briquets was made, however, and these were stronger than necessary when made with 8 per cent of pitch binder, but were rather weak with 6 per cent; 7 per cent would probably make a satisfactory briquet. When tested in a kitchen range, the briquets did not burn with satisfactory results, but after being broken into three or four pieces a better fire was obtained than with a good grade of anthracite coal of egg size. The briquets made a satisfactory fuel for an open-grate fire and for large heating stoves. They had a peculiar property of "banking," owing to the high percentage of ash in them (22.5 per cent), and a fire could be kept over night with them, even in a small kitchen range.

Briquetting test.

	Test 291.		Test 291.
Size as used:		Drop test (1-inch screen):	
Over $\frac{1}{4}$ inch..... per cent..	0.0	Held..... per cent..	16.0
$\frac{1}{4}$ inch to $\frac{1}{2}$ inch..... do....	11.9	Passed..... do.....	84.0
$\frac{1}{2}$ inch to $\frac{3}{4}$ inch..... do....	32.8	Tumbler test (1-inch screen):	
$\frac{3}{4}$ inch to 1 inch..... do....	24.3	Held..... do.....	28.0
Through $\frac{1}{4}$ inch..... do....	31.0	Passed..... do.....	72.0
Details of manufacture:		Fines through 10-mesh sieve,	
Machine used.....	Renfrow No. 2	per cent.....	94.1
Briquetting temperature, °F..	187	Weather test:	
Binder.....		Time exposed..... days..	102
Kind used.....	W. G. P.	Condition.....	B
Laboratory No.....	5940	Absorption test:	
Amount..... per cent..	6	Time immersed..... days..	31
Weight of—		Water absorbed..... per cent..	12.76
Fuel briquetted..... pounds..	22,000	Average for first 4 days..... do....	1.63
Briquet, average..... do....	0.746	Specific gravity (apparent).....	1.299
Moisture in briquet mixture, per cent.....	7.35		

Chemical analyses of culm from Merrimac, Va., and of briquets made.

	Raw fuel.	Test 291.	Pitch binder.
Laboratory No.....	5938	6114	5940
Proximate:			
Moisture..... per cent..	2.05	2.07	
Volatile matter..... do....	6.58	9.93	
Fixed carbon..... do....	58.22	65.48	
Ash..... do....	33.15	22.52	
Sulphur..... do....	.46	.52	
Ultimate:			
Hydrogen..... do....	2.43	3.04	
Carbon..... do....	59.18	68.13	
Nitrogen..... do....	.65	1.00	
Oxygen..... do....	3.43	4.30	

Extraction analyses.

	Pitch.	Fuel.	Briquets.
		J-14.	Test 291.
Laboratory No.....	5940	5938	6114
Air-drying loss..... per cent..		1.60	1.10
Extracted by CS ₂ (as received)..... do..	96.60	.06	5.19
Pitch in briquet by analyses..... do..			5.32
Heat value..... B. t. u..	16,780	9,688	11,598

WASHINGTON NO. 2.

Bituminous lump coal from Roslyn, Kittitas County, on the Northern Pacific Railway, was designated Washington No. 2, and a part (12,000 pounds) of the sample was used in briquetting test 125.

Steaming test 412 was made of briquets from this briquetting test.

Briquetting tests.

	Test 125.	Test 125.
Size as shipped.....	Lump.	Lump.
Size as used:		
Over $\frac{1}{2}$ inch..... per cent..	2.4	2.4
$\frac{1}{2}$ inch to $\frac{3}{4}$ inch..... do..	11.0	11.0
$\frac{3}{4}$ inch to $\frac{1}{2}$ inch..... do..	19.6	19.6
$\frac{1}{2}$ inch to $\frac{1}{4}$ inch..... do..	26.2	26.2
Through $\frac{1}{4}$ inch..... do..	40.8	40.8
Details of manufacture:		
Machine used.....	English.	Renfrow No. 1.
Binder—		
Kind used.....	W. G. P.	W. G. P.
Laboratory No.....	3410	3410
Amount used..... per cent..	6.5 and 7	6, 6.5, 7
Weight of—		
Fuel briquetted..... pounds..	(a)	(a)
Briquet, average..... do..	3.24	0.46
Heat value per pound:		
Fuel as received..... B. t. u..	12,586	
Briquets..... do..	12,890	
Binder..... do..	16,478	
Drop test (1-inch screen):		
Held by..... per cent..	b 87.4	
Passed..... do..	12.6	
Weather test:		
Time exposed..... days..	214	214
Condition.....	5.5 per cent C, others B.	6.5 per cent C, 7 and 7.5 per cent B.

a Total weight of both English and Renfrow briquets made was 12,000 pounds.

b Drop test made with 6 per cent briquets.

Chemical analyses of coal and briquets.

	Fuel.	Briquets.		Fuel.	Briquets.
		Test 125.			Test 125.
Laboratory No.....	3098		Ultimate:		
Proximate:			Hydrogen..... per cent..	4.96	4.86
Moisture..... per cent..	3.16	2.66	Carbon..... do..	71.62	73.02
Volatile matter..... do..	36.49	37.39	Nitrogen..... do..	1.23	1.48
Fixed carbon..... do..	48.06	48.89	Oxygen..... do..	9.09	8.88
Ash..... do..	12.28	11.06			
Sulphur..... do..	.38	.39			

Extraction analyses.

	Pitch.	Fuel.	Briquets, test 125.
		Wash No. 2.	
Laboratory No.	3410	3068	
Air-drying loss.....per cent.		1.30	0.20
Extract by CS ₂ (as received).....do.	79.98	1.24	6.89
Pitch in briquet by analysis.....do.			7.19
Heat value.....B. t. u.	16,478	12,586	12,890

WEST VIRGINIA NO. 3.^a

A sample of run-of-mine bituminous coal from near Morgantown, Monongalia County, on the Morgantown & Kingwood Railroad, was designated West Virginia No. 3, and a part of the sample was subjected to briquetting tests on the laboratory press at the St. Louis plant.

A test was made with this coal in the laboratory to determine whether it was possible, under a certain pressure and temperature, to make a briquet without any binder. A briquet was obtained, which, although not perfectly satisfactory, indicated that under certain conditions, with the right pressure and temperature, briquets could be made with this coal without any binder whatever, but it is doubtful whether a briquet made of bituminous coal in this manner would possess satisfactory strength to be of commercial value.

Chemical analysis.

	Car sample.		Car sample.
Laboratory No.	1252	Ultimate:	
Proximate:		Hydrogen.....per cent..	4.85
Air-drying loss.....per cent..	1.30	Carbon.....do.	76.89
Moisture.....do.	2.29	Nitrogen.....do.	1.45
Volatile.....do.	29.86	Oxygen.....do.	5.26
Fixed carbon.....do.	57.62	Heat value.....B. t. u.	13,558
Ash.....do.	10.23		
Sulphur.....do.	1.06		

WEST VIRGINIA NO. 6.

A sample of run-of-mine semibituminous coal from Rush Run, Fayette County, on the Chesapeake & Ohio Railroad, was designated West Virginia No. 6, and a part of the sample was subjected to briquetting tests on both the English and the American machines at the St. Louis plant.

The first test made of this coal was with 5 per cent of the glossy, home-made pitch Y. The pressure used was the limit that could be obtained on the English machine, and the resultant briquets were hard, clean, and strong, and had a sharp, glossy fracture. They were in every way equal to the best briquets made on this machine. They weighed on an average 6.12 pounds each. On account of the ex-

^a See description of tests on pp. 62-63.

cellent quality of the briquets with 5 per cent of pitch, another ton was briquetted with only 3 per cent of pitch Y. Extreme pressure was applied, and the resultant briquets were unusually smooth and glossy, but not quite strong enough. They crumbled instead of cracking clean under the blows of a hammer. The briquets containing 5 per cent of pitch, when broken, made little or no waste, and had a clean fracture.

On the American machine the mixture containing 5 per cent of pitch Y made very glossy and solid eggettes, which were strong and would stand much rough handling. One surprising quality of these briquets was that when tested in the cook stove they burned with little or no caking, although the coal itself is a very good coking coal. The briquets, when exposed to the weather, were little affected and did not disintegrate appreciably. In burning they held together until entirely consumed.

This coal was also tested with the Hoffman patent binder, and the resultant eggettes were also of good quality, being hard, glossy, and clean; but in burning they caked more than those with 5 per cent of pitch.

Chemical analysis.

	Car sample.		Car sample.
Laboratory No.....	1390	Ultimate:	
Proximate:		Hydrogen..... per cent..	4.66
Air-drying loss..... per cent..	.90	Carbon..... do.....	84.16
Moisture..... do.....	1.53	Nitrogen..... do.....	1.71
Volatile matter..... do.....	21.54	Oxygen..... do.....	3.68
Fixed carbon..... do.....	71.88	Heat value..... B. t. u..	14,807
Ash..... do.....	5.05		
Sulphur..... do.....	.65		

WEST VIRGINIA—JAMESTOWN NO. 3.

The sample designated Jamestown No. 3 consisted of two cars of semibituminous run-of-mine coal from a mine working the "Thin Vein" Pocahontas bed, Davy, McDowell County, W. Va., on the Norfolk & Western Railway.

Excellent briquets were made on the English machine with this coal and 6 per cent of binder (Lab. No. 5458), and good briquets were made on the Renfrow No. 2 machine with the same coal and binder. The briquets made on the American machine (test 257) with 7 per cent of binder (Lab. No. 5563) also had a good appearance, but were not so strong as those from the other two tests.

Briquets from test 253, while warm, had a smooth surface, metallic luster, and no cracks, but were easily crushed in the hand; when cold, however, they had a fine metallic ring when struck, a firm, fine-grained surface, and broke with difficulty, with a firm, fine-grained, lustrous fracture.

Briquets from test 256, while warm, had a smooth surface, were plastic, and broke with a rough fractured surface. When cold these

briquets had rough ends, broke without slacking, and had a smooth and glossy fracture.

Briquets from test 257 hardened at a rather high temperature and fractured with difficulty. When cold these briquets had a firm, smooth surface, were somewhat brittle, and formed some slack when broken.

Briquetting tests.

	Test 253.	Test 256.	Test 257.
Size as used:			
Over $\frac{1}{4}$ inch.....per cent.	0.8		0.1
$\frac{1}{4}$ to $\frac{1}{2}$ inch.....do.	3.5		6.1
$\frac{1}{2}$ to $\frac{3}{4}$ inch.....do.	14.7		28.8
$\frac{3}{4}$ to 1 inch.....do.	27.0		29.5
Through $\frac{1}{4}$ inch.....do.	54.0		35.5
Details of manufacture:			
Machine used.....	Englah.	Renfrow No. 2.	Renfrow No. 2.
Briquetting temperature.....° F.	202	204	205
Binder—			
Kind.....	W. G. P.	W. G. P.	W. G. P.
Laboratory No.....	5458	5458	5563
Amount.....per cent.	6	6	7
Weight of—			
Fuel briquetted.....pounds.	24,800	12,000	10,000
Briquet, average.....do.	3.53	0.617	0.588
Moisture in briquet mixture.....per cent.	6.09	2.71	4.91
Drop test (1-inch screen):			
Held.....do.	76.4	62.0	37.5
Passed.....do.	23.6	38.0	62.5
Tumbler test (1-inch screen):			
Held.....do.	71.8	58.5	47.5
Passed.....do.	28.2	41.5	52.5
Fines through 10-mesh sieve.....do.	89.7	96.0	96.0
Weather test:			
Time exposed.....days.	162	154	154
Condition.....	A	B	B
Absorption test:			
Time immersed.....days.	30	21	21
Water absorbed.....per cent.	12.02	14.46	15.79
Average for first 4 days.....do.	2.04	2.71	3.12
Specific gravity (apparent).....	1.143	1.121	1.101

Chemical analyses of coal and briquets.

	Raw fuel.	Test 253.	Test 256.	Test 257.
Laboratory No.....	5459	5477	5478	5743
Proximate:				
Moisture.....per cent.	1.52	1.86	1.69	1.06
Volatile matter.....do.	15.01	17.26	18.40	17.30
Fixed carbon.....do.	76.19	75.54	74.26	76.13
Ash.....do.	7.28	5.34	5.65	5.51
Sulphur.....do.	.62	.66	.64	.66
Ultimate:				
Hydrogen.....do.	4.45	4.45	4.50	4.63
Carbon.....do.	83.78	85.46	85.31	86.06
Nitrogen.....do.	1.14	1.09	1.11	1.10
Oxygen.....do.	2.61	2.89	2.68	1.98

Extraction analyses.

	Pitches.		Fuel, J-3.	Briquets.		
				Test 253.	Test 256.	Test 257.
Laboratory No.....	5458	5563	5459	5477	5478	5743
Air-drying loss.....per cent.			0.90	1.30	1.20	0.70
Extracted by CS_2 (as received).....do.	92.92	90.33	0.47	5.03	5.28	5.86
Pitch in briquet, by analysis.....do.				4.93	5.20	6.00
Heat value.....B. t. u.	16,893	16,718	14,443	14,828	14,728	14,886

WEST VIRGINIA—JAMESTOWN NO. 5.

The sample designated Jamestown No. 5 consisted of two cars of semibituminous run-of-mine coal from a mine working the Sewell bed, 1 mile east of Sewell, Fayette County, W. Va., on the Chesapeake & Ohio Railway.

Of the two tests made on this coal, the briquets of test 258 were better than those of test 259. The warm briquets of test 258 had smooth surfaces and were difficult to fracture. The cold briquets were very hard and brittle and formed some slack when broken. The warm briquets from test 259 had smooth surfaces, were very plastic, and high in moisture; the cold briquets had a very hard, firm surface, and fractured without slacking.

Briquetting tests.

	Test 258.	Test 259.
Size as used:		
Over $\frac{1}{2}$ inch..... per cent.....	0.1	
$\frac{1}{2}$ inch to $\frac{3}{4}$ inch..... do.....	0.5	
$\frac{3}{4}$ inch to 1 inch..... do.....	27.0	
1 inch to $\frac{3}{2}$ inch..... do.....	31.0	
Through $\frac{3}{2}$ inch..... do.....	35.3	
Details of manufacture:		
Machine used.....	Renfrow No. 2	Renfrow No. 2
Briquetting temperature..... °F.....	199	200
Binder		
Kind.....	W. G. P.	W. G. P.
Laboratory No.....	5563	5453
Amount..... per cent.....	7	7
Weight of—		
Fuel briquetted..... pounds.....	136,000	2,000
Briquet, average..... do.....	0.563	0.598
Moisture in briquet mixture..... per cent.....	6.14	
Drop test (1-inch screen):		
Held..... do.....	70.0	48.5
Passed..... do.....	30.0	51.5
Tumbler test (1-inch screen):		
Held..... do.....	71.5	67.5
Passed..... do.....	28.5	32.5
Fines through 10-mesh sieve..... do.....	95.5	96.2
Weather test:		
Time exposed..... days.....	154	154
Condition.....	A	A
Absorption test:		
Time immersed..... days.....	21	21
Water absorbed..... per cent.....	14.90	15.26
Average for first 4 days..... do.....	2.52	2.62
Specific gravity (apparent).....	1.062	1.103

Chemical analyses of coal and briquets.

	Raw fuel.	Test 258.
Laboratory No.....	5453	5744
Proximate:		
Moisture..... per cent.....	3.37	1.32
Volatile matter..... do.....	21.80	24.97
Fixed carbon..... do.....	70.74	67.91
Ash..... do.....	4.09	5.80
Sulphur..... do.....	.58	.63
Ultimate:		
Hydrogen..... do.....	4.85	4.81
Carbon..... do.....	85.06	83.88
Nitrogen..... do.....	1.39	1.16
Oxygen..... do.....	3.87	3.68

Extraction analyses.

	Pitches.		Fuel, J-5.	Briquets, test 268.
Laboratory No.	5563	5458	5453	5744
Air-drying loss. per cent.			2.60	0.90
Extracted by CS_2 (as received) do.	90.33	92.92	0.51	6.62
Pitch in briquet by analysis do.				6.81
Heat value. B. t. u.	16,718	16,893	14,522	14,677

WEST VIRGINIA—JAMESTOWN NO. 6.

The sample designated Jamestown No. 6 consisted of two cars of semibituminous run-of-mine coal from a mine working the Sewell bed, Redstar Fayette County, W. Va., on the Chesapeake & Ohio Railway.

The briquets made from this coal were excellent in their physical characteristics, and when warm had smooth surfaces, sharp edges, and were sufficiently strong to be loaded on cars direct from the briquetting machine. When cold, these briquets were hard and firm, but somewhat brittle on the edges.

Briquetting test.

	Test 260.		Test 260.
Size as used:		Drop test (1-inch screen).	
Over $\frac{1}{4}$ inch. per cent.	0.8	Held. per cent.	80.2
$\frac{1}{4}$ inch to $\frac{1}{2}$ inch. do.	3.1	Passed. do.	19.8
$\frac{1}{2}$ inch to $\frac{3}{4}$ inch. do.	11.2	Tumbler test (1-inch screen):	
$\frac{3}{4}$ inch to 1 inch. do.	26.3	Held. do.	77.4
Through 1 inch. do.	54.8	Passed. do.	22.6
Details of manufacture:		Fines through 10-mesh sieve, per cent.	88.5
Machine used. English.	203	Weather test:	
Briquetting temperature. °F.		Time exposed. days.	135
Binder—		Condition.	A
Kind. W. G. P.	5563	Absorption test:	
Laboratory No.	6	Time immersed. days.	35
Amount. per cent.		Water absorbed. per cent.	13.19
Weight of—		Average for first 4 days. do.	2.24
Fuel briquetted. pounds.	41,400	Specific gravity (apparent)	1.117
Briquet, average. do.	3.52		
Moisture in briquet mixture. per cent.	5.47		

Chemical analyses of coal and briquets.

	Raw fuel.	Test 260.		Raw fuel.	Test 260.
Laboratory No.	5489	5558	Ultimate:		
Proximate:			Hydrogen. per cent.	4.74	4.96
Moisture. per cent.	2.06	2.15	Carbon. do.	81.31	85.20
Volatile matter. do.	19.39	21.87	Nitrogen. do.	1.36	1.56
Fixed carbon. do.	74.63	71.13	Oxygen. do.	7.71	2.49
Ash. do.	3.89	4.85			
Sulphur. do.	.89	.81			

Extraction analyses.

	Pitch.	Fuel, J-5.	Briquets, test 260.
Laboratory No.	5563	5489	5558
Air-drying loss. per cent.		1.50	1.50
Extracted by CS_2 (as received) do.	90.33	0.36	5.32
Pitch in briquet by analysis do.			5.51
Heat value. B. t. u.	16,718	14,783	14,715

WEST VIRGINIA—JAMESTOWN NO. 7.

The sample designated Jamestown No. 7 consisted of three cars of semibituminous run-of-mine coal from a mine working the Sewell bed, Derryhale, Fayette County, W. Va., on the Chesapeake & Ohio Railway.

The briquets made on the Renfrow No. 2 machine from this coal with 7 per cent of binder (Lab. No. 5563) stood the cohesion tests better than those made on the English machine with 6 per cent of the same binder. Both were excellent briquets, gave a fine metallic ring when struck, had smooth, firm surfaces, broke with difficulty, and had firm and glossy fractures.

Briquetting tests.

	Test 261.	Test 262.
Size as used:		
Over $\frac{1}{2}$ inch..... per cent..	3.0	0.3
$\frac{1}{2}$ inch to $\frac{3}{4}$ inch..... do.	6.9	6.0
$\frac{3}{4}$ inch to $\frac{7}{8}$ inch..... do.	10.8	25.1
$\frac{7}{8}$ inch to $\frac{15}{16}$ inch..... do.	22.4	29.3
Through $\frac{15}{16}$ inch..... do.	56.9	39.3
Details of manufacture:		
Machine used.....	English.	Renfrow No. 2.
Briquetting temperature..... ° F.	198	193
Binder—		
Kind.....	W. G. P.	W. G. P.
Laboratory No.....	5563	5563
Amount..... per cent..	6	7
Weight of—		
Fuel briquetted..... pounds..	43,200	38,000
Briquet, average..... do.	5.53	0.617
Moisture in briquet mixture..... per cent..	5.48	4.71
Drop test (1-inch screen):		
Held..... do.	69.3	71.0
Passed..... do.	30.7	29.0
Tumbler test (1-inch screen):		
Held..... do.	74.0	80.5
Passed..... do.	26.0	19.5
Fines through 10-mesh sieve..... do.	88.6	98.2
Weather test:		
Time exposed..... days..	132	132
Condition.....	A	B
Absorption test:		
Time immersed..... days..	35	11
Water absorbed..... per cent..	11.09	12.21
Average for first 4 days..... do.	2.14	2.85
Specific gravity (apparent).....	1.139	1.029

Chemical analyses of coal and briquets.

	Raw fuel.	Test 261.	Test 262.
Laboratory No.....	5501	5583	5559
Proximate:			
Moisture..... per cent..	2.00	2.70	0.93
Volatile matter..... do.	17.57	20.28	18.52
Fixed carbon..... do.	74.10	72.74	74.65
Ash..... do.	6.33	4.28	5.90
Sulphur..... do.	.91	.81	.81
Ultimate:			
Hydrogen..... do.	4.59	4.77	4.69
Carbon..... do.	83.03	86.07	84.46
Nitrogen..... do.	1.31	1.49	1.51
Oxygen..... do.	3.68	2.44	2.56

Extraction analyses.

	Pitch.	Fuel, J-7.	Briquets.	
			Test 261.	Test 262.
Laboratory No.....	5563	5501	5583	5559
Air-drying loss.....per cent..		1.40	2.10	0.30
Extract by CS ₂ (as received).....do..	90.33	.40	4.99	6.56
Pitch in briquet by analysis.....do..			5.12	6.85
Heat value.....B. t. u..	16,718	14,465	14,886	14,717

WEST VIRGINIA—JAMESTOWN NO. 8.

The sample designated Jamestown No. 8 consisted of two cars of semibituminous run-of-mine coal from a mine working the Quinnimont (Fire Creek) bed, Lawton, Fayette County, W. Va., on the Chesapeake & Ohio Railway.

This coal was briquetted with binders as follows: Tests 263, 264, and 270, with flour binder alone; test 271, with water-gas pitch alone; and the remaining tests, 265, 266, 267, 268, and 269, with various mixtures of flour and water-gas pitch.

All of the briquets made with flour alone disintegrated very rapidly on exposure to the weather, and one sample (from test 270) crumbled to slack when handled after being immersed in water for 24 hours. The briquets made with 1 per cent of flour binder were not strong enough to be satisfactory, but all of the other briquets made with flour alone or with a mixture of flour and water-gas pitch stood the cohesion tests better than the briquets that had only water-gas pitch for a binder. A comparison of tests 267, 268, and 269 with test 271 shows that in the first three tests the drop and tumbler percentages were higher than in the fourth test. All briquets containing any flour soon became covered with a green mold when stored in a damp place and lost strength, but those stored in a dry place developed no mold, and even after eight months were seemingly as good as when first made. In making briquets with flour alone, some difficulty was experienced from the briquets sticking to the dies of the press, but it is probable that when the conditions are better understood this trouble may be obviated. The briquets from test 268 were stronger than those from any other of the nine tests made of this coal, but those from test 269 were almost as strong. The briquets made with 7 per cent of water-gas pitch alone (test 271) were of excellent quality, but were somewhat sticky when hot, although they were loaded into the car direct from the machine with very little breakage. No samples were taken for sizing tests and chemical analyses from the briquets made in tests 263 to 270, inclusive, as all these tests were of preliminary nature.

Briquetting tests.

	Test No.				
	263.	264.	270.	265.	266.
Details of manufacture:					
Machine used.....	Renfrow No.2	English.	Renfrow No.2	English.	English.
Briquetting temperature, °F.....	198	201		201	
Binder—					
Kind.....	Flour.	Flour.	Flour.	Flour.	Flour.
Laboratory No.....	6110	6110	6110	W. G. P. Flour 6110	W. G. P. Flour 6110
Amount..... per cent.....	1.0	2.0	4	W. G. P. 5939 1 flour.	W. G. P. 5939 2 flour.
Weight of—				3 W. G. P.	2 W. G. P.
Fuel briquetted..... lbs.....	500	500	500	500	500
Briquet, average..... do.....	0.558	3.55	0.595	3.47	3.55
Moisture in briquet mixture, per cent.....			7.29		
Drop test (1-inch screen):					
Held..... per cent.....	65.5	60.2	79.5	60.2	76.1
Passed..... do.....	34.5	39.8	20.5	39.8	23.9
Tumbler test (1-inch screen):					
Held..... do.....	31.9	56.0	80.0	67.2	75.6
Passed..... do.....	68.1	44.0	11.00	32.8	24.4
Fines through 10-mesh sieve..... per cent.....	92.7	92.0	96.0	94.0	89.2
Weather test:					
Time exposed..... days.....	132	132	132	132	132
Condition.....	E	E	E	C	D
Absorption test:					
Time immersed..... days.....	8	9	1	8	4
Water absorbed, per cent.....	5.94	9.76	10.60	9.31	10.29
Average for first four days..... per cent.....	1.53	2.44	(a)	2.31	2.57
Specific gravity (apparent).....	1.222	1.184	1.079	1.168	1.157

	Test No.			
	267.	268.	269.	271.
Size as used:				
Over ½ inch..... per cent.....				0.0
¾ inch to 1 inch..... do.....				9.3
1 inch to 1½ inch..... do.....				35.5
1½ inch to 2 inch..... do.....				29.2
Through 2 inch..... do.....				28.0
Details of manufacture:				
Machine used.....	Renfrow No. 2	Renfrow No. 2	Renfrow No. 2	Renfrow No. 2
Briquetting temperature, °F.....				196
Binder—				
Kind.....	Flour.	Flour.	Flour.	W. G. P.
Laboratory No.....	W. G. P. Flour 6110	W. G. P. Flour 6110	W. G. P. Flour 6110	5941
Amount..... per cent.....	2 flour.	2 flour.	2 flour.	7
Weight of—	2 W. G. P.	3 W. G. P.	2 W. G. P.	
Fuel briquetted..... pounds.....	500	500	500	81,250
Briquet, average..... do.....	0.562	0.588	0.562	0.633
Moisture in briquet mixture, per cent.....		6.29	5.72	5.21
Drop test (1-inch screen):				
Held..... do.....	84.0	87.0	84.5	46.5
Passed..... do.....	16.0	13.0	15.5	53.5
Tumbler test (1-inch screen):				
Held..... per cent.....	81.0	91.0	88.0	58.0
Passed..... do.....	19.0	9.0	12.0	42.0
Fines through 10-mesh sieve, do.....	96.7	96.7	97.0	95.5
Weather test:				
Time exposed..... days.....	132	132	132	127
Condition.....	E	B	C	A
Absorption test:				
Time immersed..... days.....	4	5	4	12
Water absorbed..... per cent.....	11.47	8.09	16.56	15.31
Average for first 4 days..... do.....	2.87	2.00	4.14	3.56
Specific gravity (apparent).....	1.150	1.196	1.067	1.100

* Sample crumbled after one day's immersion.

Chemical analyses of semibituminous coal from Lawton, W. Va., and of flour binder.

	Raw fuel.	Test 271.	Flour binder.
Laboratory No.	5575	5713	6110
Proximate:			
Moisture..... per cent.	2.80	3.28	13.30
Volatile matter..... do.	17.10	18.50	72.30
Fixed carbon..... do.	74.80	73.44	13.50
Ash..... do.	5.30	4.78	.90
Sulphur..... do.	.86	.84	.10
Ultimate:			
Hydrogen..... do.	4.75	4.75	
Carbon..... do.	85.07	85.06	
Nitrogen..... do.	1.44	1.11	
Oxygen..... do.	2.41	3.28	

Extraction analyses.

	Binders.			Fuel, J-8.	Briquets, test 271.
	Pitch.	Flour.	Pitch.		
Laboratory No.	5941	6110	5939	5575	5713
Air-drying loss..... per cent.				2.20	2.90
Extracted by CS ₂ (as received)..... do.	98.44		98.13	0.52	4.80
Pitch in briquets by analysis..... do.				1.11	4.38
Heat value..... B. t. u.	16,978	6,998	16,781	14,701	14,584

WEST VIRGINIA—JAMESTOWN NO. 9.

The sample designated as Jamestown No. 9 consisted of two cars of semibituminous run-of-mine coal from a mine working the Sewell bed, Winona, Fayette County, W. Va., on the Chesapeake & Ohio Railway.

The coal was briquetted with three different kinds of pitch binder. The briquets from test 274 made with water-gas pitch (Lab. No. 5939) were the strongest of the three. All three tests made excellent briquets, which gave a good metallic ring when knocked together, had smooth, hard surfaces, were broken with difficulty, and had firm and glossy fractured surfaces. No analyses of briquets from tests 273 and 274 were made.

Briquetting tests.

	Test No.		
	272.	273.	274.
Size as used:			
Over $\frac{1}{2}$ inch..... per cent.	0.0		
$\frac{1}{2}$ inch to $\frac{3}{4}$ inch..... do.	7.9		
$\frac{3}{4}$ inch to $\frac{1}{2}$ inch..... do.	22.9		
$\frac{1}{2}$ inch to $\frac{1}{4}$ inch..... do.	30.5		
Through $\frac{1}{4}$ inch..... do.	37.1		
Details of manufacture:			
Machine used.....	Renfrow No. 2	Renfrow No. 2	Renfrow No. 2
Briquetting temperature..... °F.	195	198	198
Binder—			
Kind.....	W. G. P.	W. G. P.	W. G. P.
Laboratory No.....	5941	5563	5939
Amount..... per cent.	7	7	7
Weight of—			
Fuel briquetted..... pounds.	46,000	2,000	2,000
Briquet, average..... do.	0.625	0.602	0.610

Briquetting tests—Continued.

	Test No.		
	272.	273.	274.
Moisture in briquet mixture.....per cent.	5.65		
Drop test (1-inch screen):			
Held.....do.	57.5	71.0	73.0
Passed.....do.	42.5	29.0	27.0
Tumbler test (1-inch screen):			
Held.....do.	64.0	77.0	83.5
Passed.....do.	36.0	23.0	16.5
Fines through 10-mesh sieve.....do.	95.1	96.7	96.6
Weather test:			
Time exposed.....days.	125	52	125
Condition.....do.	A	A	A
Absorption test:			
Time immersed.....days.	12	21	13
Water absorbed.....per cent.	16.66	13.02	13.14
Average for first 4 days.....do.	3.49	2.75	2.79
Specific gravity (apparent).....do.	1.072	1.118	1.117

Chemical analyses of coal and briquets.

	Raw fuel.	Test 272.
Laboratory No.....	5709	5746
Proximate:		
Moisture.....per cent.	3.96	2.64
Volatile matter.....do.	23.21	25.29
Fixed carbon.....do.	67.41	67.61
Ash.....do.	5.40	4.46
Sulphur.....do.	.66	.65
Ultimate:		
Hydrogen.....do.	5.01	5.00
Carbon.....do.	82.85	84.57
Nitrogen.....do.	1.40	1.14
Oxygen.....do.	4.43	4.05

Extraction analyses.

	Pitches.			Fuel, J-9.	Briquets, test 272.
Laboratory No.....	5941	5563	5939	5709	5745
Air-drying loss.....per cent.				3.00	2.10
Extracted by CS ₂ (as received).....do.	98.44	90.33	98.13	0.69	4.77
Pitch in briquet by analysis.....do.					4.18
Heat value.....B. t. u.	16,978	16,718	16,781	14,238	14,699

WEST VIRGINIA—JAMESTOWN NO. 10.

The sample designated as Jamestown No. 10 consisted of two cars of semibituminous run-of-mine coal from a mine working the Beckley bed, Stanaford, Raleigh County, W. Va., on the Chesapeake & Ohio Railway.

Test 275 gave satisfactory briquets with 6 per cent of water-gas pitch (Lab. No. 5941). The briquets had smooth, firm surfaces, sharp edges, were broken with difficulty, and had firm and glossy fracture surfaces.

Briquetting test.

	Test 275.		Test 275.
Size as used:		Drop test (1-inch screen):	
Over $\frac{1}{2}$ inch..... per cent.	4.5	Held..... per cent..	33.7
$\frac{1}{2}$ inch to $\frac{3}{4}$ inch..... do.	16.0	Passed..... do.	66.3
$\frac{3}{4}$ inch to 1 inch..... do.	28.8	Tumbler test (1-inch screen):	
1 inch to $\frac{3}{8}$ inch..... do.	25.4	Held..... do.	55.0
Through $\frac{3}{8}$ inch..... do.	25.3	Passed..... do.	45.0
Details of manufacture:		Fines through 10-mesh sieve..... do.	90.0
Machine used.....	English.	Weather test:	
Briquetting temperature..... °F.	182	Time exposed..... days..	120
Binder—		Condition.....	A
Kind.....	W. G. P.	Absorption test:	
Laboratory No.....	5941	Time immersed..... days..	21
Amount..... per cent.	6	Water absorbed..... per cent..	8.75
Weight of—		Average for first 4 days..... do.	2.07
Fuel briquetted..... pounds.	53,550	Specific gravity (apparent).....	1.206
Briquet, average..... do.	3.83		
Moisture in briquet mixture per cent..	6.45		

Chemical analyses of coal and briquets.

	Raw fuel.	Test 275.
Laboratory No.....	5719	5746
Proximate:		
Moisture..... per cent..	2.14	3.83
Volatile matter..... do.	16.83	18.69
Fixed carbon..... do.	71.91	69.03
Ash..... do.	9.12	8.45
Sulphur..... do.	1.20	.91
Ultimate:		
Hydrogen..... do.	4.57	4.63
Carbon..... do.	81.34	81.32
Nitrogen..... do.	1.44	1.24
Oxygen..... do.	2.10	3.07

Extraction analyses.

	Pitch.	Fuel, J-10.	Briquets, test 275.
Laboratory No.....	5941	5719	5746
Air-drying loss..... per cent..		1.40	3.40
Extracted by CS ₂ (as received)..... do.	98.44	0.64	5.07
Pitch in briquet by analysis..... do.			4.53
Heat value..... B. t. u..	16,978	14,024	13,871

WEST VIRGINIA—JAMESTOWN NO. 11.

The sample designated as Jamestown No. 11 consisted of two cars of semibituminous run-of-mine coal from a mine working the Beckley bed, West Raleigh, Raleigh County, W. Va., on the Chesapeake & Ohio Railway.

In test 276 excellent briquets were made from this coal with 7 per cent of water-gas pitch (Lab. No. 5941).

In test 277 six tons of briquets was made on the English machine, with 3 per cent of flour for a binder. The briquets were strong even after standing for three months in a damp place, and briquets kept in a dry place for seven months appeared as good as when first made.

After exposure to the weather for six weeks in the wintertime, a sample of the briquets could be handled and was still serviceable, so that it would seem that flour may be considered a satisfactory binder when the briquets made from it are not exposed to the weather for more than two months.

In test 289, in order to compare flour and pitch binders, 6 tons of this coal was briquetted with 6 per cent of water-gas pitch (Lab. No. 5940); an excellent briquet was obtained. A comparison of the briquets from this test with those made with flour binder (test No. 277) shows that the flour briquets were stronger as indicated by physical tests but did not resist the effects of weathering as well as the pitch briquets.

In test 290 a mixture of 50 per cent of Jamestown No. 11 and 50 per cent of anthracite coal (buckwheat size), designated as Jamestown No. 18, was made to determine whether fine anthracite could be improved by briquetting with a bituminous coal. A very satisfactory briquet was obtained which had a firm, smooth surface, was hard to break, and had firm and glossy fracture surfaces.

Briquetting tests.

	Test 276.	Test 277.	Test 289.	Test 290. ^a
Size as used:				
Over $\frac{1}{2}$ inch.....per cent..	0.1		0.4	5.7
$\frac{1}{2}$ inch to $\frac{3}{4}$ inch.....do..	7.4		4.2	17.2
$\frac{3}{4}$ inch to $\frac{1}{2}$ inch.....do..	29.9		23.2	19.6
$\frac{1}{2}$ inch to $\frac{1}{4}$ inch.....do..	27.7		39.6	18.9
Through $\frac{1}{4}$ inch.....do..	34.9	11.41	32.6	38.6
Details of manufacture:				
Machine used.....	Renfrow No. 2	English.	English.	English.
Briquetting temperature.....° F..	190	185		191
Binder—				
Kind.....	W. G. P.	Flour.	W. G. P.	W. G. P.
Laboratory No.....	5941	6110	5940	5940
Amount.....per cent..	7	3	6	6
Weight of—				
Fuel briquetted.....pounds..	42,000	12,000	12,750	12,350
Briquet, average.....do..	0.610	3.78	3.77	4.15
Moisture in briquet mixture.....per cent..	4.60	11.41	10.10	8.48
Drop test (1-inch screen):				
Held.....do..	70.0	77.2	68.2	68.3
Passed.....do..	30.0	22.8	31.8	31.7
Tumbler test (1-inch screen):				
Held.....do..	77.5	60.7	72.0	73.5
Passed.....do..	22.5	39.3	28.0	21.5
Fines through 10-mesh sieve.....do..	94.0	88.3	88.2	87.5
Weather test:				
Time exposed.....days..	120	120	102	102
Condition.....	A	D	A	A
Absorption test:				
Time immersed.....days..	9	5	31	31
Water absorbed.....per cent..	13.45	4.43	9.82	9.85
Average for first 4 days.....do..	3.25	1.05	1.21	1.31
Specific gravity (apparent).....	1.117	1.239	1.161	1.247

^a Jamestown No. 11, 50 per cent; Jamestown No. 18, 50 per cent.

Chemical analyses of coal and briquets.

	Raw fuel.	Test 276.	Test 277.	Test 289.	Test 290.
Laboratory No.....	5718	5756	6111	6112	6113
Proximate:					
Air-drying loss..... per cent..	1.50	2.30	3.20	0.80	2.20
Moisture..... do.....	2.15	2.73	3.83	1.61	3.42
Volatile matter..... do.....	15.06	13.27	15.77	20.30	13.03
Fixed carbon..... do.....	75.46	73.56	74.50	71.98	75.93
Ash..... do.....	7.33	5.44	5.90	6.11	7.62
Sulphur..... do.....	.90	.66	.69	.79	.63
Ultimate:					
Hydrogen..... do.....	4.58	4.54	4.13	4.00	3.50
Carbon..... do.....	84.09	85.15	82.59	82.58	83.01
Nitrogen..... do.....	1.53	1.36	.88	1.47	1.50
Oxygen..... do.....	1.24	2.68	5.55	4.94	3.45
Extract by CS_2 (as received)..... do.....	.59	5.21		4.63	5.25
Pitch in briquet by analysis..... do.....		4.72		4.21	5.12
Heat value..... B. t. u.....	14,391	14,630	14,186	14,584	13,921

WEST VIRGINIA—JAMESTOWN NO. 12.

The sample designated Jamestown No. 12 consisted of one car of semibituminous coal from a mine working Pocahontas No. 3 bed, Switchback, McDowell County, W. Va., on the Norfolk & Western Railway.

This coal was briquetted first with 6 per cent of water-gas pitch (Lab. No. 5941), and such excellent briquets were obtained that test 279 was made to see whether 5 per cent of the same binder would make a satisfactory briquet. The briquet with 5 per cent of pitch was found to be better than that containing 6 per cent. Both tests made excellent briquets, which had firm, smooth surfaces, were broken with difficulty, and had firm, close-grained, and shiny fracture surfaces.

Briquetting tests.

	Test 278.	Test 279.
Size as used:		
Over $\frac{1}{2}$ inch..... per cent..	5.6	5.6
$\frac{1}{2}$ inch to $\frac{3}{4}$ inch..... do.....	18.1	18.1
$\frac{3}{4}$ inch to 1 inch..... do.....	33.8	33.8
1 inch to $\frac{3}{2}$ inch..... do.....	23.2	23.2
Through $\frac{3}{2}$ inch..... do.....	19.3	19.3
Details of manufacture:		
Machine used.....	Renfrow No. 2.	Renfrow No. 2.
Briquetting temperature..... ° F..	195	194
Binder—		
Kind.....	W. G. P.	W. G. P.
Laboratory No.....	5941	5941
Amount..... per cent..	6	5
Weight of—		
Fuel briquetted..... pounds..	38,000	16,000
Briquets, average..... do.....	0.676	0.709
Moisture in briquet mixture..... per cent..	5.03	
Drop test (1-inch screen):		
Held..... do.....	59.0	66.5
Passed..... do.....	41.0	33.5
Tumbler test (1-inch screen):		
Held..... do.....	72.0	78.5
Passed..... do.....	28.0	21.5
Fines through 10-mesh sieve..... do.....	90.9	93.2
Weather test:		
Time exposed..... days..	137	137
Condition.....	A	A
Absorption test:		
Time immersed..... days..	5	4
Water absorbed..... per cent..	11.73	12.18
Average for first 4 days..... do.....	2.91	3.05
Specific gravity (apparent).....	1.261	1.248

Chemical analyses of coal and briquets.

	Raw fuel. ^a	Test 278.
Laboratory No.	5706	5856
Proximate:		
Moisture.....per cent.	3.01	1.71
Volatile matter.....do.	15.94	13.96
Fixed carbon.....do.	78.81	68.36
Ash.....do.	4.74	15.97
Sulphur.....do.	.44	.42
Ultimate:		
Hydrogen.....do.		3.87
Carbon.....do.		74.41
Nitrogen.....do.		.78
Oxygen.....do.		4.26

^a Analysis of mine sample; no car sample taken.*Extraction analyses.*

	Pitch.	Fuel, J-12.	Briquets, test 278.
Laboratory No.	5941	5706	5856
Air-drying loss.....per cent.		2.40	1.10
Extracted by CH ₂ (as received).....do.	98.44	0.17	5.74
Pitch in briquet by analysis.....do.			5.67
Heat value.....B. t. u.	16,978	14,715	12,938

WEST VIRGINIA—JAMESTOWN NO. 13.

The sample designated Jamestown No. 13 consisted of three cars of semibituminous run-of-mine coal from a mine working the Pocahontas No. 3 bed, Ennis, McDowell County, W. Va., on the Norfolk & Western Railway.

This coal was briquetted on both the English and the Renfrow No. 2 machines. The briquets were shipped from the plant for locomotive tests without any samples being retained, so no physical tests could be made of the briquets. In appearance, just after making, the briquets made on the English machine were very satisfactory and had firm, smooth surfaces.

The locomotive steaming tests are briefly described under the heading "Locomotive-boiler Tests."

Briquetting tests.

	Test 286.	Test 287.
Details of manufacture:		
Machine used.....	English.	Renfrow No. 2
Binder—		
Kind.....	W. G. P.	W. G. P.
Laboratory No.	5940	5940
Amount.....per cent.	6	6
Weight of fuel briquetted.....pounds.	82,960	160,000

Chemical analyses of coal and briquets.

	Raw fuel.	Test 286.
Laboratory No.....	5829	5048
Proximate:		
Moisture.....per cent..	3.67	3.10
Volatile matter.....do....	15.09	17.08
Fixed carbon.....do.....	74.69	73.11
Ash.....do.....	6.55	6.71
Sulphur.....do.....	.46	.52
Ultimate:		
Hydrogen.....do.....	4.06	4.16
Carbon.....do.....	84.84	83.33
Nitrogen.....do.....	.92	1.00
Oxygen.....do.....	2.84	4.05

Extraction analyses.

	Pitch.	Fuel, J. 13.	Briquets, test 286.
Laboratory No.....	5040	5829	5048
Air-drying loss.....per cent..		2.80	2.00
Extracted by CS ₂ (as received).....do....	96.60	.34	5.17
Pitch in briquet by analysis.....do.....			5.02
Heat value.....B. t. u..	16,780	14,290	14,290

WEST VIRGINIA—JAMESTOWN NO. 15.

The sample designated Jamestown No. 15 consisted of several cars of semibituminous run-of-mine coal from a mine working the Sewell bed, Minden, Fayette County, W. Va., on the Chesapeake & Ohio Railway, and was tested at the Norfolk plant.

In test 280, owing to the presence of an excessive amount of moisture in this coal, it was difficult to make satisfactory briquets on the English machine. Water cracks formed, and the briquets had a porous surface which was easily rubbed off. They were easily broken, forming considerable slack. As the flowing point of the binder used in this test was 194° F., the briquetting temperature of 187° F. was not high enough to soften the pitch sufficiently to obtain the best results.

In test 281 the briquets made on the Renfrow No. 2 machine were probably stronger than those of test 280, made on the English machine, but the sample saved for the physical tests was mislaid, making it impossible to apply tests. The briquets were hard and showed no water cracks, although the coal was very wet.

The briquets made were delivered to the Chesapeake & Ohio Railway for trial under locomotive boilers. For results of the locomotive steaming tests see pages 114 and 115.

Briquetting tests.

	Test 280.	Test 281.
Size as used:		
Over $\frac{1}{2}$ inch.....per cent..	0.1	0.0
$\frac{1}{2}$ inch to $\frac{3}{4}$ inch.....do..	8.9	5.1
$\frac{3}{4}$ inch to $\frac{1}{2}$ inch.....do..	27.1	28.6
$\frac{1}{2}$ inch to $\frac{1}{4}$ inch.....do..	27.0	32.8
Through $\frac{1}{4}$ inch.....do..	36.9	33.5
Details of manufacture:		
Machine used.....	English.	Renfrow No. 2
Briquetting temperature.....°F.	187	190
Binder—		
Kind.....	W. G. P.	W. G. P.
Laboratory No.....	5941	5941
Amount.....per cent..	6	6
Weight of—		
Fuel briquetted.....pounds	65,800	300,000
Briquets, average.....do..	3.57	0.720
Moisture in briquet mixture.....per cent..	7.04	6.87
Drop test (1-inch screen):		
Held.....do..	19.6	-----
Passed.....do..	80.4	-----
Tumbler test (1-inch screen):		
Held.....do..	48.8	-----
Passed.....do..	51.2	-----
Fines through 10-mesh sieve.....do..	88.9	-----
Weather test:		
Time exposed.....days	137	125
Condition.....	B	B
Absorption test:		
Time immersed.....days	5	11
Water absorbed.....per cent..	7.87	12.01
Average for first 4 days.....do..	1.95	2.92
Specific gravity (apparent).....	1.185	1.079

Chemical analyses of coal and briquets.

	Raw fuel.	Test 280.	Test 281.
Laboratory No.....	5774	5857	5859
Proximate:			
Moisture.....per cent..	3.80	1.09	3.21
Volatile matter.....do..	21.32	18.70	19.49
Fixed carbon.....do..	67.80	73.60	72.67
Ash.....do..	7.58	6.61	4.63
Sulphur.....do..	.68	.96	.68
Ultimate:			
Hydrogen.....do..	4.56	4.63	4.63
Carbon.....do..	82.14	81.77	84.69
Nitrogen.....do..	1.36	1.21	1.31
Oxygen.....do..	3.46	4.75	3.90

Extraction analyses.

	Pitch.	Fuel, J-15.	Briquets.	
			Test 280.	Test 281.
Laboratory No.....	5941	5774	5857	5859
Air-drying loss.....per cent..	-----	3.00	0.40	2.50
Extracted by CS_2 (as received).....do..	98.44	0.42	5.84	5.77
Pitch in briquet by analysis.....do..	-----	-----	5.22	5.46
Heat value.....B. t. u..	16,978	14,009	14,548	14,573

WEST VIRGINIA—JAMESTOWN NO. 16.

The sample designated Jamestown No. 16 consisted of semibituminous run-of-mine coal from West Virginia.

The coal was briquetted with four different kinds of water gas pitch binders, 6 per cent of each being used. Seemingly the best briquets were those of test 285, although those of test 282 were practically as good. Both of these tests were made on the English machine. Test 283, made on the Renfrow No. 2 machine, produced satisfactory briquets, but the briquets made on the English machine with water-gas pitch in test 284 (Lab. No. 5939) were not satisfactory, as they did not stand the physical tests well.

Tests 283 and 285 made briquets with smooth, firm, fine-grained surfaces; they broke with difficulty, and had firm and hard fracture surfaces.

Test 282 made good briquets as far as strength was concerned, but their surfaces were rough and coarse-grained, and their edges rubbed off somewhat easily. They broke with difficulty, and the fracture surface was coarse, firm, and dull-black in color.

Test 284 furnished very poor briquets, the surfaces of which were rubbed off easily, owing either to too small a percentage of pitch or to too low a briquetting temperature; the briquets were easily broken, and the fractured surfaces were rough and easily rubbed off.

The entire sample was made into briquets and delivered to the Atlantic Coast Line Railway Co. for locomotive tests, coal from the same locality being regularly used by the engines on that road.

A brief description of the locomotive steaming tests and results is to be found under the heading "Locomotive-boiler Tests."

Briquetting tests.

	Test 282.	Test 283.	Test 284.	Test 285.
Size as used:				
Over $\frac{1}{2}$ inch.....per cent..	5.0	0.1		
$\frac{1}{2}$ inch to $\frac{3}{4}$ inch.....do....	16.0	14.4		
$\frac{3}{4}$ inch to $\frac{1}{2}$ inch.....do....	31.4	36.0		
$\frac{1}{2}$ inch to $\frac{1}{4}$ inch.....do....	26.3	25.9		
Through $\frac{1}{4}$ inch.....do....	21.3	23.6		
Details of manufacture:				
Machine used.....	English.	Renfrow No. 2.	English.	Renfrow No. 2.
Briquetting temperature.....°F..	174	189		
Binder:				
Kind.....	W. G. P.	W. G. P.	W. G. P.	W. G. P.
Laboratory No.....	4879	5941	5939	5940
Amount.....per cent..	6	6	6	6
Weight of—				
Fuel briquetted.....pounds..	52,048	66,000	30,600	37,400
Briquets, average.....do....	4.10	0.641	3.83	3.68
Moisture in briquet mixture.....per cent..	9.55	7.55		8.40
Drop test (1-inch screen):				
Held.....do....	77.3	35.5	39.2	78.8
Passed.....do....	22.7	64.5	60.8	21.2
Tumbler test (1-inch screen):				
Held.....do....	82.7	56.0	47.8	82.8
Passed.....do....	17.2	44.0	52.2	17.2
Fines through 10-mesh sieve.....do....	89.0	90.1	90.1	89.5
Weather test:				
Time exposed.....days..	137	125	118	118
Condition.....	B	B	B	A
Absorption test:				
Time immersed.....days..	31	31	31	31
Water absorbed.....per cent..	9.79	10.55	15.98	11.23
Average for first 4 days.....do....	0.89	1.12	1.82	1.13
Specific gravity (apparent).....	1.206	1.126	1.112	1.143

Chemical analyses of coal and briquets.

	Raw fuel.	Test 282.	Test 283.	Test 284.	Test 285.
Laboratory No.....	5825	5946	5945	5947	5853
Proximate:					
Moisture..... per cent.	4.89	2.21	2.27	2.06	8.40
Volatile matter..... do.	29.63	30.97	32.56	31.65	
Fixed carbon..... do.	57.74	52.97	54.62	58.74	
Ash..... do.	7.74	13.85	8.65	7.55	
Sulphur..... do.	.72	.89	.85	.90	
Ultimate:					
Hydrogen..... do.	4.69	4.27	4.67	4.69	
Carbon..... do.	77.63	71.89	77.73	78.27	
Nitrogen..... do.	1.27	1.11	1.26	1.27	
Oxygen..... do.	7.51	7.66	6.63	7.24	

Extraction analyses.

	Pitches.				Fuel, J-16.	Briquets.			
						Test 282.	Test 283.	Test 284.	Test 285.
Laboratory No.....	4879	5939	5940	5941	5825	5946	5945	5947	5853
Air-drying loss..... per cent.					3.40	1.20	1.40	1.20	7.00
Extracted by CS_2 (as re- ceived)..... per cent.	94.50	98.13	98.60	98.44	0.68	5.43	6.39	5.10	
Pitch in briquet by analysis, per cent.....						5.06	5.84	4.54	
Heat value..... B. t. u.	16,806	16,781	16,780	16,978	13,480	12,790	13,754	13,939	

WEST VIRGINIA—JAMESTOWN NO. 17.

The sample designated Jamestown No. 17 consisted of semibituminous run-of-mine coal from the Pocahontas No. 3 bed of West Virginia, and was tested at the Norfolk plant.

Eighty tons of this coal was briquetted for a special test on the battleship *Connecticut* of the United States Navy, and as it was shipped without any samples being retained, no data exist on which to make a report.

Briquetting tests.

	Test 283.
Details of manufacture:	
Machine used.....	American.
Binder—	
Kind.....	W. G. P.
Laboratory No.....	5940
Amount..... per cent..	6
Weight of fuel briquetted..... pounds..	160,000

Chemical analyses.

	Raw fuel.		Raw fuel.
Laboratory No.....	5832	Ultimate:	
Proximate:		Hydrogen..... per cent..	4.28
Moisture..... per cent.	3.16	Carbon..... do.	87.01
Volatile matter..... do.	15.90	Nitrogen..... do.	1.00
Fixed carbon..... do.	76.73	Oxygen..... do.	2.77
Ash..... do.	4.31		
Sulphur..... do.	.52		

Extraction analyses.

	Pitch.	Fuel, J-17.
Laboratory No.	5940	5832
Air-drying loss. per cent.		2.80
Extracted by CS_2 (as received)..... do.	96.00	
Heat value..... B. t. u.	16,780	14,746

WYOMING NO. 1.

A sample of black lignite coal (over a 5-inch screen) from near Sheridan, Sheridan County, on the Burlington system, was designated Wyoming No. 1, and a part of the sample was subjected to briquetting tests on the English machine at the St. Louis plant.

This coal was a black, pitchy-looking lignite, which contained 23 per cent of water. It was first tried without drying, 9 per cent of pitch H being used. Judging from the resulting briquets there was a slight excess of pitch, but on account of the large proportion of water in the lignite the pitch seemingly failed to adhere to the grains of lignite. The fresh briquets did not stand handling well, and when broken open the grains of lignite seemed to be coated with moisture, which prevented the pitch from adhering to them. Many of the briquets were porous, not being sufficiently pressed, and they were also brittle. Some of them, however, were fairly good. When broken and burned in the stove they stood up well in the fire and burned better than any lignite briquets that had thus far been made. They weighed on an average 6.31 pounds each.

Another ton of this lignite was ground in the Williams mill and run through the drier, but this operation removed only a part of the water, so that the briquets still contained considerable moisture. This material was briquetted on the English machine with 8 per cent of pitch H. A minimum amount of steam was used, which did not contain any excess of water, and a maximum pressure was applied. As received from the machine, many of the briquets showed perpendicular cracks, due to excessive pressure. The hot briquets did not stand handling, the individual grains of the lignite seeming to be wet and not cohering at all. Many of them would crack badly under their own weight. On breaking open the hot briquets the grains of lignite seemed to move about as if alive. This lack of cohesion continued until the pitch became rather hard and could not longer be pressed in the machine. Variation in the pressure of steam used as the mixture passed through the pug mill had no effect on the cohesion of the briquets. On cooling, however, these briquets became very hard and strong and broke with a sharp fracture and with only a small waste. In burning they did not fall to pieces, as did the other lignite briquets, but burned satisfactorily. They were very light, weighing only 5.85 pounds each, and had a specific gravity of 0.98. The specific gravity of the coal was only 1.16. The crushing strength of these briquets was 9,600 pounds, or 339 pounds to the square inch.

Another ton of this coal was tested with a soft, tough asphalt, B6, from Casper, Wyo. As had been shown by laboratory tests, this asphalt could not be used alone on the English machine, and as a result the lignite was briquetted with 5 per cent of asphalt B6, 2½ per cent of rosin, and 1 per cent of lime that had previously been hydrated. The mixture worked perfectly in the machine, and the hot briquets were firm and easily handled, but the proportion of binder was not sufficient, and the briquets were dry and not strong. Another reason for this result was that during the two crushings and dryings of the lignite the grains had become of a nearly uniform size (about 10-mesh fineness), so that the briquets were very porous in spite of the maximum pressure used. In burning, the briquets fell to pieces. They weighed on an average 5.40 pounds each. A combination of this coal and asphalt can undoubtedly be obtained that will briquet satisfactorily.

A further test of this lignite was made with 7 per cent of a mixture composed of 20 parts asphalt and 1 part rosin, with 1 part slaked lime. The coal had been previously dried. The briquets were poor and disintegrated badly in the fire. This coal was again tried with the same binder without previous drying. The resultant briquets, though better physically, did not have so much cohesion in the fire as the first. It is probable that a partial drying of this lignite would give the best results.

A supply of Wyoming lignite received from Casper, Wyo., was briquetted without previous drying with 8 per cent of a mixture composed of 2 parts of Wyoming asphalt to 1 part of rosin and 1 per cent of slaked lime. The resultant briquet stood up in the fire fairly well, although it disintegrated a little. With 7 per cent of the mixture and 1 per cent of slaked lime the briquet was practically the same; with 6 per cent of the mixture and 1 per cent of slaked lime the briquet was crumbly; and with 7 per cent of the mixture without lime the briquet was physically a little tougher, but disintegrated more in the fire than when the lime was used. On increasing the percentage of lime and using 7 per cent of the mixture and 2 per cent of slaked lime a briquet was obtained that looked much like those made with the 7 to 1 combination and was not any stronger.

Chemical analysis of subbituminous coal from Sheridan, Wyo.

	Car sample.		Car sample.
Laboratory No.	1479	Ultimate:	
Proximate:		Hydrogen.....per cent..	5.01
Air-drying loss.....per cent..	6.00	Carbon.....do.....	70.97
Moisture.....do.....	22.63	Nitrogen.....do.....	1.33
Volatile matter.....do.....	35.68	Oxygen.....do.....	16.13
Fixed carbon.....do.....	37.19	Heat value.....B. t. u..	9,734
Ash.....do.....	4.50		
Sulphur.....do.....	.59		

WYOMING NO. 6.

Subbituminous run-of-mine coal from 3 miles west of Kemmerer, Lincoln County, 2 miles from the Union Pacific Railroad, was designated Wyoming No. 6, and a part (24,000 pounds) of the sample was used in briquetting test 134. Steaming test 419 was made of the briquets from the above test.

In test 134 the details were as follows: Size of coal used: Over one-fourth inch, 2.4 per cent; one-tenth inch to one-fourth inch, 9.2 per cent; one-twentieth inch to one-tenth inch, 22.2 per cent; one-fortieth inch to one-twentieth inch, 26 per cent; through one-fortieth inch, 40.2 per cent; kind of binder, water-gas pitch; laboratory number, 3486.

English briquets, with an average weight of 3.18 pounds, and made at 179° F., with 5, 6, 7, 8, and 10 per cent of binder, would not cohere; all the briquets could be crushed in the hand. In the weathering test they were exposed 180 days; the condition of those with 5 and 6 per cent of binder was D; of those with 7 and 8 per cent binder, E.

Drop test (14-inch screen).

	Percentage of binder.			
	5	6	7	8
Held.....	35.4	61.3	74	79
Passed.....	64.6	38.7	26	21

Renfrow No. 1 machine briquets made at 149° F., average weight 0.42 pound, with 7, 8, and 10 per cent binder, were not improved by an increase in the percentage of pitch; all of the briquets were poor and could be crushed in the hand. In the weathering test they were exposed 180 days when their condition was E.

Chemical analyses of coal and briquets.

	Car sample.	Test 134.		Car sample.	Test 134.
Laboratory No.....	3390		Ultimate:		
Proximate:			Hydrogen..... per cent..	5.31	4.99
Moisture..... per cent..	19.00	15.57	Carbon..... do.....	73.31	74.59
Volatile matter..... do....	36.64	38.74	Nitrogen..... do.....	1.21	1.03
Fixed carbon..... do....	41.24	42.87	Oxygen..... do.....	15.72	15.35
Ash..... do.....	3.12	2.82			
Sulphur..... do.....	.49	.50			

Extraction analyses.

	Pitch.	Fuel.	Briquets.
		Wyo. No. 6.	Test 134.
Laboratory No.....	3486	3390	
Air-drying loss..... per cent..		11.30	4.90
Extracted by CS ₂ (as received)..... do....	85.57	0.83	5.82
Pitch in briquet by analysis..... do....			6.13
Heat value..... B. t. u..	16,407	10,307	10,994

MISCELLANEOUS NO. 5.

A sample of coke breeze furnished by a gas company was used in briquetting test 152 and, in mixture with Tennessee No. 4, in tests 150 and 151. The data of briquetting tests Nos. 150 and 151 are included under the discussion of Tennessee No. 4.

In test 152 the size of coal used was as follows: Over one-fourth inch, 2.8 per cent; one-tenth inch to one-fourth inch, 11 per cent; one-twentieth inch to one-tenth inch, 21.2 per cent; one-fortieth inch to one-twentieth inch, 30.4 per cent; through one-fortieth inch, 34.6 per cent. The briquets were very soft when warm, showing nearly 25 per cent breakage in falling from the delivery belt of the machine and in handling with coke forks. They could not be piled satisfactorily while warm. When cold they had a rough, hard surface, with a coating of brown from the dies, and broke without crumbling, giving a rough fracture surface.

Briquetting tests.

	Test 152.		Test 152.
Details of manufacture:		Tumbler test (1-inch screen):	
Machine used.....	Renfrow No. 1	Held.....per cent..	72.5
Briquetting temperature, ° F..	149	Passed.....do.....	27.5
Binder—		Fines through 10-mesh sieve,	
Kind.....	W. G. P.	per cent.....	90.0
Laboratory No.....	3410	Weather test:	
Amount.....per cent..	8	Time exposed.....days..	198
Weight of—		Condition.....	A
Fuel briquetted..pounds..	20,000	Absorption test:	
Briquet, average....do....	0.437	Time immersed.....days..	15
Drop test (1-inch screen):		Water absorbed.....per cent..	20.6
Held.....per cent.....	60.5	Average for first 5 days...do....	2.64
Passed.....do.....	39.5	Specific gravity (apparent).....	1.030

Chemical analyses of coal and briquets.

	Fuel.	Test 152.		Fuel.	Test 152.
Laboratory No.....	3386	3385	Fixed carbon.....per cent..	73.25	72.06
Moisture.....per cent..	10.72	5.01	Ash.....do.....	11.95	12.24
Volatile matter.....do....	4.08	10.70	Sulphur.....do.....	1.02	1.06

Extraction analyses.

	Pitch.	Fuel.	Briquets, test 152.
Laboratory No.....	3410	3386	3385
Air-drying loss.....per cent..		8.80	3.10
Extracted by CS ₂ (as received).....do....	79.98	0.59	6.74
Pitch in briquet by analyses.....do....			7.75
Heat value.....B. t. u..	16,478	11,086	12,119

MISCELLANEOUS NO. 9.

Coke breeze from a plant in Madison, Ill., was designated Miscellaneous No. 9 and was used in briquetting tests 247 and 249; also, in mixture with Pennsylvania No. 18, in briquetting tests 238, 239, 240, and 248. The briquets from test 247 were used in a special cupola.

test. A description and data from tests 238, 239, 240, and 248 are found under the heading "Pennsylvania No. 18."

In test 247 the size of coal used was as follows: Over one-fourth inch, 2.2 per cent; one-tenth inch to one-fourth inch, 8 per cent; one-twentieth inch to one-tenth inch, 15 per cent; one-fortieth inch to one-twentieth inch, 25.6 per cent; through one-fortieth inch, 49.2 per cent. Attempts to briquet coke breeze with unslaked lime as a binder were unsatisfactory. Briquets were made by the addition of pitch to the lime, and showed when warm characteristics similar to those of the briquets made in test 249. The cold briquets, however, were much harder and continued to increase in hardness for several weeks. The hardened briquets had a smooth outer surface and a rough, hard fracture surface, with firm edges.

In test 249 the size of coal used was as follows: Over one-fourth inch, 1.6 per cent; one-tenth inch to one-fourth inch, 7.6 per cent; one-twentieth inch to one-tenth inch, 15.8 per cent; one-fortieth inch to one-twentieth inch, 25.8 per cent; through one-fortieth inch, 49.2 per cent. The characteristics of these briquets were similar to those of the briquets discussed in the report of test 248. The briquets had a coating of bronze from the dies, which showed perceptibly the erosive effect of the coke. Satisfactory briquets were made with 8 per cent of binder, and no increase in cohesion was observed when more were used.

Briquetting tests.

	Test 247. ^a	Test 249.
Details of manufacture:		
Machine used.....	Renfrow No. 1.	Renfrow No. 1.
Briquetting temperature..... ^b F.	176	158
Binder—		
Kind.....	(^b)	W. G. P.
Laboratory No.....	4879	4879
Amount..... per cent..	8 of each.	8
Weight of—		
Fuel briquetted..... pounds..	6,000	2,000
Briquet, average..... do.	0.642	0.520
Drop test (1-inch screen):		
Held..... per cent..	66.0	67.0
Passed..... do.	34.0	33.0
Tumbler test (1-inch screen):		
Held..... do.	86.5	79.5
Passed..... do.	13.5	20.5
Fines through 10-mesh sieve..... do.	65.0	81.0
Absorption test:		
Time immersed..... days..	10	23
Water absorbed..... per cent..	10.0	18.8
Average for first—		
6 days..... do.	1.50	
5 days..... do.		2.60
Specific gravity (apparent).....	1.287	1.178

^a The briquets made in this test were used in a special cupola test.

^b Water-gas pitch and unslaked lime, 8 per cent each.

Chemical analyses of coal and briquets.

	Raw fuel.	Tests 247 and 249.		Raw fuel.	Tests 247 and 249.
Laboratory No.....	4763	4827	Fixed carbon..... per cent..	75.50	58.55
Moisture..... per cent..	1.95	2.61	Ash..... do.	21.06	28.18
Volatile matter..... do.	.89	10.71	Sulphur..... do.	.66	.98

Extraction analyses.

	Pitch.	Fuel.	Briquets, tests 247 and 249.
Laboratory No.	4879	4763	4927
Air-drying loss. per cent.		1.30	1.30
Extracted by CS ₂ do.	94.80	0.15	7.22
Pitch in briquet by analysis. do.			7.50
Heat value. B. t. u.	16,806	10,870	9,325

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Bulletin 59

DEPARTMENT OF THE INTERIOR

BUREAU OF MINES

JOSEPH A. HOLMES, DIRECTOR

INVESTIGATIONS OF DETONATORS AND ELECTRIC DETONATORS

BY

CLARENCE HALL

AND

SPENCER P. HOWELL



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INVESTIGATIONS OF DETONATORS AND ELECTRIC DETONATORS.

By CLARENCE HALL and SPENCER P. HOWELL.

INTRODUCTION.

Among the more important factors involved in the use of high explosives in blasting operations is the means employed to bring about the detonation of the charge. When flame is applied to high explosives many of them may burn if not confined; but all of them when burning under certain conditions of confinement may detonate. Detonation may also be effected by mechanical means, such as frictional impact caused by a blow or by rubbing between surfaces. By this means, however, the full effect of the explosive charge may not be developed, so that a partial detonation, often accompanied by the burning of the explosive, results.

When nitroglycerin was first used it was fired by the application of flame, but considerable difficulty was experienced in exploding it with certainty and in obtaining uniform results. In 1864 Alfred Noble, a Swedish engineer, discovered that nitroglycerin could be surely and completely detonated by exploding in contact with it a small quantity of an initiatory explosive. Mercury fulminate was the substance then found capable of producing the best results. There are many other fulminates and other substances that will produce complete detonation of commercial "high" explosives, but detonators or electric detonators containing mercury fulminate as the characteristic ingredient are still almost exclusively used in this country.

The term "detonator" is used in the publications of the Bureau of Mines to designate what the miner calls a "blasting cap"—a copper capsule containing a small quantity of some detonating compound that is ignited by a fuse. The term "electric detonator" is applied to a blasting cap that is similar except for being ignited by means of a small wire which is heated to incandescence or fused by the passage of an electric current.

One of the conditions prescribed by the Bureau of Mines for a permissible explosive ^a is that it shall be fired by a detonator, or preferably an electric detonator, having a charge equivalent to that of the standard detonator used at the Pittsburgh testing station. A further

^a Permissible explosives have a short, quick flame and are intended especially for use in coal mines containing inflammable gases or dusts. (See Miners' Circular 6, Bureau of Mines.)

requirement is that this charge shall consist by weight of 90 parts of mercury fulminate and 10 parts of potassium chlorate (or their equivalents).

At the request of a manufacturer of permissible explosives, an investigation was undertaken by the bureau to determine the relative strength of detonators and electric detonators having different compositions. The tests of electric detonators herein reported were conducted by H. F. Braddock, junior chemist; J. W. Koster, J. E. Tiffany, junior mining engineers; and A. S. Crossfield, junior explosives chemist, at the Pittsburgh testing station of the bureau. Similar tests of detonators were not conducted because it was believed that the results would not show sufficient variation to warrant such tests. It is hoped that the conclusions drawn from the tests made will be of service to those using explosives by enabling them to select the grade of detonator or electric detonator that will insure the most effective results. The conclusions are given in this bulletin, which is published by the Bureau of Mines as one of a series of publications dealing with the testing of explosives and the precautions that should be taken to increase safety and efficiency in the use of explosives in mining operations.

The results of the experiments described in this bulletin show that the average percentage of failures of explosives to detonate was increased more than 20 per cent when the lower grades of electric detonators were used instead of No. 6 electric detonators, and was increased more than 50 per cent when these lower grades were used instead of No. 8 electric detonators. It is noteworthy, however, that when sensitive explosives, such as 40 per cent strength ammonia dynamite (p. 33), were tested under conditions ideal for detonation, the same energy was developed irrespective of the electric detonator used. When tests were made with a less sensitive explosive, such as a 40 per cent strength ammonia dynamite containing nitrosubstitution compounds (p. 32), the energy developed increased with the grade of the electric detonator used. For example, the average efficiency of four different explosives was increased 10.4 per cent when a No. 6 electric detonator was used instead of a No. 4 electric detonator, and 14.9 per cent when a No. 8 electric detonator was used (see tabulation on p. 45). The results of the tests emphasize the importance of using explosives in a fresh condition, but as fresh explosives can not always be had in mining work, strong detonators should be used in order to offset any deterioration of explosives from age.

The results obtained substantiate the following conclusions: (1) That for any particular manufacturer's detonators or electric detonators the explosive efficiency increases with their grade, and (2) that the four No. 6 electric detonators, of different makes, tested have practically the same explosive efficiency as, and each is considered equivalent to, the Pittsburgh testing station standard No. 6 electric

detonator for use with permissible explosives in coal mines when the No. 6 grade is prescribed.

PRELIMINARY CONSIDERATIONS.

Methods for determining the strength of detonators or electric detonators by mechanical effects may be classed as either direct or indirect. The direct method comprises those tests in which the mechanical effect of the detonators or electric detonators is determined. The indirect method comprises those tests in which the mechanical effect of the explosives with which the detonators or electric detonators are used is determined. The direct method offers the advantage of simplicity, and usually of cheapness, but may lead to grave inaccuracies unless checked by mechanical effects indirectly determined. The discussion under the heading, "Tests Previously Used to Determine the Strength of Detonators" illustrates this.

The indirect method of determining the mechanical effects of explosives, or the energy developed by them, approximates practical conditions and offers an accurate means for determining the relative efficiency of detonators and electric detonators in bringing about complete detonation of commercial "high" explosives.

As all direct methods of testing detonators are therefore dependent on the indirect method for verification, the first experiments undertaken were to determine the relative strength of electric detonators indirectly by comparing the energy developed by different commercial explosives when fired with different grades of electric detonators. Afterwards tests were made by determining the relative strength of electric detonators by direct means, and a test was devised that, although not entirely satisfactory, gave results that approximated more closely those established by the indirect tests.

Detonating explosives develop their energy in the most efficient way when fired with detonators or electric detonators that completely detonate or explode them. Obviously, if the detonation be incomplete, a part of the potential energy of the explosive will not be released, and the loss of energy will be proportional to the percentage of the charge that did not detonate. In blasting operations an incomplete detonation is not only a menace to safety, by reason of the possible explosion of the unexploded part of the charge and of the harmful gaseous products resulting from the blast, but in many cases it acts like an underloaded shot and performs little, if any, useful work.

If an explosive is in a fresh condition and is sensitive to detonation and no obstacles are present to hinder its detonation, then any detonator effective enough to cause its complete detonation will develop its full energy.

In practice, however, conditions ideal for detonation rarely and perhaps never exist, because the commercial explosives are somewhat insensitive to detonation or because they may have deteriorated by

aging before use. Furthermore, crimped paper ends of the cartridges, loose material in the drill hole, air spaces between the cartridges, or cartridges of too small diameter may hinder detonation.

An explosive is said to age when any physical or chemical change during storage affects its sensitiveness, its uniformity, or its stability. Such changes are usually caused by the temperature and the humidity of the air or by sunlight, and even gravity may have an important effect. If the explosive is placed in the sunlight, it may become unstable. If a cartridge of dynamite is subjected to a temperature above 90° F., gravity may cause the segregation of nitroglycerin in the lower end or side of the cartridge. If nitroglycerin explosives are subjected to temperatures alternately above and below 52° F., the nitroglycerin tends to segregate in the cartridge. These conditions affect the uniformity of the explosives. If explosives, especially those containing ammonium nitrate or other hygroscopic salts, are subjected to a moist atmosphere they tend to absorb moisture. If the temperature is less than 52° F., nitroglycerin explosives other than low-freezing ones may freeze, and low-freezing explosives will freeze at a temperature less than 35° F. Recently, there have been placed on the market nitroglycerin explosives, styled nonfreezing explosives, that are declared by the manufacturers to remain unfrozen when the temperature falls as low as 0° F. Both moisture and freezing affect the sensitiveness of explosives.

The results of preliminary tests indicated that it would be impossible to discriminate between commercial electric detonators by determining the energy developed by explosives used with them unless the explosives were insensitive to detonation or tested under conditions which would simulate their use under actual mining conditions; consequently the authors, in testing electric detonators indirectly, used explosives in an insensitive condition. This was done by using those that were naturally insensitive, such as an explosive of class 1,^a subclass b; by using explosives in cartridges having small diameters, such as the 20 per cent "straight" nitroglycerin dynamite in cartridges of $\frac{1}{4}$ -inch diameter; in an aged condition, such as the 35 per cent strength gelatin dynamite two years old; in a frozen condition, such as 40 per cent strength gelatin dynamite; and, in the case of ammonia dynamite, by the addition of water.

The apparatus used in the experimental study were the Mettegang recorder, small lead blocks, and Trauzl lead blocks. The results of the tests differentiated the electric detonators in two ways. In the first place the electric detonator either did or did not cause the detonation of the explosives. In tabulating such results the number of detonations is expressed as the percentage of the number of trials. Only the tests of those explosives were considered in which at least one failure to detonate occurred and in which detonation occurred in

^a For an explanation of classification see p. 12.

at least one trial. In the second place, those trials in which detonation occurred were used as a basis of comparing the relative explosive efficiency of the electric detonators. The results of only those tests in which each of the electric detonators of the series caused detonation were recorded. The results are expressed in percentages of explosive efficiency as compared with the Pittsburgh testing station standard No. 6 electric detonator. This electric detonator offered the advantage of being included in both groups tested—the Pittsburgh testing station standard and the four No. 6 electric detonators.

THEORY OF DETONATION.

A short discussion of the theory of detonation as presented by Berthelot* is necessary in order that a better interpretation of the experiments herein reported can be made. The theory is called the "explosive-wave" theory, and it has been generally accepted because all detonation phenomena can be best explained by it. In order to analyze the propagation of an explosive wave, the wave is considered as a recurring cycle of released and transformed energy with four phases, as follows: Mechanical to calorific, calorific to chemical (the phase in which the potential energy of the explosive material is released), chemical to calorific, and calorific to mechanical.

This cycle can best be readily understood by indicating how the explosive wave is propagated through a cylindrical file of a homogeneous explosive without the loss of enough energy to interrupt propagation.

1. *Transformation of mechanical energy to calorific energy.*—When an explosive detonates a part of the mechanical energy of a layer of the explosive is converted instantly into heat energy in the adjacent layer by reason of the impact of molecules. The efficiency of this conversion is low—certainly less than 50 per cent—as the movement of the molecules is radial and they are only partly confined by the layer of explosive in the file. The mechanical energy that is not converted into heat energy exerts pressure on the confining medium and thus becomes the vehicle through which work is accomplished. There is good reason for believing that the thickness of the layer of explosive that enters into the first phase of the cycle varies with the physical properties of the explosive material, principally with its elasticity and partly with the velocity of the molecules that are in molecular vibration. The less elastic the explosive material and the greater the velocity of the molecules the thinner the layer, and hence the more times the cycle will recur in a unit length of the explosive material.

2. *Transformation of calorific energy to chemical energy.*—Some of the calorific energy of the layer is used to overcome the chemical stability of the explosive material, which may vary widely, and thus release the potential energy of the layer; the rest of the calorific

* Berthelot, M., *Explosives and their power*, 1892, pp. 82-113.

energy is used to accelerate and reinforce the chemical action. The layer of explosive by this time is developing a tremendous kinetic energy as expressed in phase 3.

3. *Transformation of chemical energy to calorific energy.*—All commercial explosives develop heat on detonation. This phase is different from the others because each of those represents some kind of kinetic energy derived entirely from the preceding phase, and consequently no one of them can have more kinetic energy than the preceding phase is capable of transferring. The conversion in this phase is complete because all the potential energy released becomes kinetic energy, which is largely calorific energy.

4. *Transformation of calorific energy to mechanical energy.*—A simple statement of this phase is that the larger volume of gases then formed from the layer of explosives is in an extremely active state of molecular vibration and that these molecules are then manifesting their energy as mechanical energy. The efficiency of conversion of calorific energy to mechanical energy is high because the conversion is very rapid and radiation and conduction losses are correspondingly small.

DETONATION OF HIGH EXPLOSIVES.

All methods used to initiate the explosive wave, or to detonate high explosives, involve the application of heat. If heat be applied directly by means of a flame such as is produced by a fuse, squib, or electric igniter, or by a spark or an incandescent solid, and the explosive be of the first order, or directly explosive, such as mercury fulminate or iodide of nitrogen, then detonation is sure and effective. If, however, the explosive be of the second order, or indirectly explosive, such as dynamite, permissible explosives, trinitrotoluene, or guncotton, then detonation, especially complete detonation, does not usually occur; hence the direct application of heat is not a sure and effective means of producing detonation.

If heat, such as is produced by the physical resistance of the explosive to a blow or impact, be applied indirectly to high explosives, then any sufficient blow or impact will cause detonation; that is, it will initiate the four-phase energy cycle, or explosive wave.

Because the impact produced by detonators is extremely quick, and their mercury-fulminate composition has a high density and releases considerable kinetic energy, the force of the impact is instantly converted into heat which is applied to a thin layer of the explosive material, thereby overcoming the chemical stability of that layer and initiating the explosive wave. Experience and investigation has proved this means of producing the detonation of explosives, those not too insensitive, to be both sure and effective; hence one is not surprised to learn that detonators are universally used.

As the mercury-fulminate composition of detonators is an explosive of the first order it may be detonated by fire, and hence fuse may

be used in connection with them. Fuse is made of a uniform outside diameter and detonators are made of a uniform inside diameter such that the fuse fits snugly into them. In using fuse, it is cut square across and inserted into the detonator until it gently touches the fulminate mixture and then the detonator is crimped on the fuse.

Similarly a detonator may be fired by means of a small platinum wire embedded into the priming composition and brought to incandescence or fused by the passage of an electric current. (See figs. 1 and 4.) The priming composition may be simply an easily inflamed material such as loose guncotton, a match composition, an explosive of the first order such as mercury fulminate, or a mercury-fulminate composition. The priming composition is placed in the detonator directly above and in contact with the main charge. The platinum bridge is attached at each end to an insulated wire; the two wires, called the legs, pass through the plug and the filling, and are connected by leading wires to the source of the electric current. When a detonator is fitted with means of firing by an electric current it is called an electric detonator. Electric detonators are particularly adapted to shot firing in fiery mines, or to the simultaneous firing of several charges. They are also adapted to any purpose for which detonators may be used, and as their use offers a greater assurance of safety they are growing in favor.

ELECTRIC DETONATORS TESTED.

The electric detonators tested were designated as the Pittsburgh testing station standard No. 3, No. 4, No. 5, No. 6, No. 7, and No. 8, the Western Coast No. 6, the special No. 6, and the foreign No. 6. For brevity the expression Pittsburgh testing station standard is abbreviated in this paper to P. T. S. S.

The P. T. S. S. No. 3 electric detonators were made at the testing station from No. 3 detonators. A cross-sectional view of one of these electric detonators is shown in figure 1. The priming charge consisted of 0.02 gram of dry, loose guncotton directly above and in contact with the compressed charge. The sulphur plug, the insulated-wire legs, and the platinum bridge were so placed that the bridge was embedded in the loose guncotton. Then the molten sulphur was poured over the plug until the cap was filled.

As detonators in this country are made of a uniform inside diameter of 0.220 inch and electric detonators of a uniform inside diameter of 0.260 inch, the P. T. S. S. No. 3 electric detonators are smaller in diameter than all the others except the special No. 6 electric detonators which were also assembled at the Pittsburgh testing station. It was impossible to procure No. 3 electric detonators in the open market, as their manufacture has recently been discontinued.

The priming charge used in the No. 3, the No. 5, and the No. 7 electric detonators consisted of loose guncotton; that in the No. 4,

the No. 6, and the No. 8 electric detonators was commercially pure mercury fulminate.

The Western Coast No. 6 and the foreign No. 6 electric detonators were used as received. The special No. 6 electric detonator was made at the testing station in the same manner as the P. T. S. S. No. 3. The primer of the western coast No. 6 was loose gun-cotton; that of the foreign No. 6 was a mixture of picric acid and chlorate of potash. The foreign No. 6 was so called because the detonator was imported, but the priming charge, sulphur plug, and wires were assembled by a manufacturer in this country.

These electric detonators are representative of all the electric detonators commercially used in the United States.

The P. T. S. S. No. 4, No. 5, No. 6, No. 7, and No. 8 were used as received from the manufacturers. Because of the seemingly erratic results of tests with the P. T. S. S. No. 5 electric detonators, attention is called to the fact that they were from 3 to 3½ years old when used, and that although the sulphur plug protected the fulminating composition somewhat, they were not in first-class condition.

EXPLOSIVES USED IN THE TESTS.

The explosives used in the tests are enumerated below; they included certain permissible explosives and different grades of commercial dynamites. Explosives designated as permissible by the bureau are grouped in four classes.* Class 1, ammonium-nitrate explosives, includes all explosives in which the characteristic material is ammonium nitrate. The class is divided into two subclasses: Subclass *a*, including every ammonium-nitrate explosive that contains a sensitizer that is itself an explosive, and subclass *b*, including every ammonium-nitrate explosive that contains a sensitizer that is not in itself an explosive. Class 2, hydrated explosives, includes all explosives in which salts containing water of crystallization are the characteristic materials. Class 3, organic-nitrate explosives, includes all explosives in which the characteristic material is an organic nitrate other than nitroglycerin. Class 4, nitroglycerin explosives, includes all explosives in which the characteristic material is nitroglycerin.

The permissible explosives used in the tests were as follows: Sample 1, sample 2, and sample 3 of an explosive of class 1, subclass *a*; sample 1 and sample 2 of an explosive of class 1, subclass *b*; and an explosive of class 4.

The commercial grades of dynamites used were a 20 per cent "straight" nitroglycerin dynamite; a 40 per cent strength ammonia dynamite (containing nitrosubstitution compounds); a 40 per cent strength ammonia dynamite; a 35 per cent strength gelatin dynamite (2 years old); a 35 per cent strength gelatin dynamite (3 years old); and a 40 per cent strength gelatin dynamite.

* See Miners' Circular 6, Bureau of Mines, 1912, p. 16.

The results of physical examination of the above-mentioned explosives were as follows:

Results of physical examination of explosives used in tests.

Class and grade of explosives.	Diameter of cartridge.	Length of cartridge.	Average weight.	Cartridges re-dipped.	Apparent specific gravity of cartridge by sand.	Color.	Consistence.
Class 1, subclass e (sample 1).	1½	8	160	No.	1.01	Corn	Granular and fibrous; fine; soft; dry; slightly cohesive.
Class 1, subclass e (sample 2).	1½	8	174	Yes.	1.09	do	Do.
Class 1, subclass e (sample 3).	1½	8	227	Yes.	.93	Mauve	Powdered; very fine; soft; dry; not cohesive.
Class 1, subclass b (sample 1).	1½	8	277	Yes.	.88	Corn	Powdered; very fine; very dry; very soft; not cohesive.
Class 1, subclass b (sample 2).	1½	8	278	Yes.	.88	do	Do.
Class 4.....	1½	8	166	No.	1.00	do	Granular and fibrous; soft; fine; dry; slightly cohesive.
20 per cent "straight" nitroglycerin dynamite.	1	8	103	No.	1.18	do	Do.
40 per cent strength ammonia dynamite (containing nitrosubstitution compounds).	1½	7½	226	Yes.	1.34	do	Fibrous; very fine; dry; soft; slightly cohesive.
40 per cent strength ammonia dynamite.	1½	8	241	Yes.	1.43	Drab.....	Granular; fine; dry; soft; slightly cohesive.
35 per cent strength gelatin dynamite (2 years old).	1½	7½	265	No.	1.63	Corn	Gelatinous; fine; wet; soft; moderately cohesive.
35 per cent strength gelatin dynamite (3 years old).	1½	7½	339	No.	1.66	do	Do.
40 per cent strength gelatin dynamite.	1½	7½	295	No.	1.60	Drab.....	Do.

Certain of the different explosives used in the tests were analyzed, with results as follows:

Results of analyses of certain explosives used in tests.

Constituent.	Kind of explosives.					
	20 per cent "straight" nitroglycerin dynamite. ^a	40 per cent strength ammonia dynamite containing nitrosubstitution compounds). ^b	40 per cent strength ammonia dynamite. ^c	35 per cent strength gelatin dynamite (3 years old). ^b	35 per cent strength gelatin dynamite (3 years old). ^b	40 per cent strength gelatin dynamite. ^c
Moisture.....	1.30	1.93	0.88	1.80	5.86	1.47
Nitroglycerin.....	19.54	16.28	21.00	29.03	28.10	30.70
Nitrocellulose.....		4.97				
Nitrocellulose.....				.68	1.17	.88
Sodium nitrate.....	62.09	47.14	46.04	48.62	52.20	54.27
Ammonium nitrate.....		18.78	18.89			
Wood pulp.....	15.22		5.45		5.55	5.58
Wood pulp and crude fiber.....		2.84		2.15		
Calcium carbonate.....	1.95		1.44	1.13		
Zinc oxide.....		.62	.59		1.07	1.62
Sulphur.....		2.84	4.85	4.83	4.58	3.08
Starch.....		3.79		11.47		
Vaseline.....		.81				
Paraffin.....					1.24	
Resin.....					.28	
Total.....	100.00	100.00	100.00	100.00	100.00	100.00

^a Analyst, W. C. Cope.

^b Analyst, A. L. Hyde.

^c Contains 1.04 per cent sodium chloride.

TESTS PREVIOUSLY USED TO DETERMINE STRENGTH OF DETONATORS AND ELECTRIC DETONATORS.

Six principal tests have been used previously to determine the strength of detonators or electric detonators. They are as follows:

1. *Weight of charge.*—Ever since it was observed that certain explosives would not always detonate with a certain weight of charge of mercury fulminate or mercury-fulminate composition and that these same explosives would always detonate if the weight of charge in the detonator was increased, it has been customary to vary the charge in the detonators and to consider the weight of the charge to be an indication of the strength of the detonator. There are several grades of detonators, and they are designated by the charge of fulminate composition contained in them.

Bigg-Wither^a arranged the following table, which was published in 1900:

Weight of charge in different grades of detonators.

Grade No.	Charge per detonator.	
	Grams.	Grains.
1	0.30	4.6
2	.40	6.2
3	.54	8.3
4	.65	10.0
5	.80	12.3
6	1.00	15.4
6½	1.25	19.2
7	1.50	23.1
8	2.00	30.9

It is to be noted that in 1900 there was no great variation in the composition of detonators. There is no indication that the relation between the effectiveness of the detonator and the weight of the charge was other than directly as the first-power function.

2. *Deformation or penetration of lead or iron plates.*^b—Guttman^c states: "One of the oldest and most frequently used tests for measuring the power of caps (used only with ordnance) consisted of exploding them on a lead or iron plate resting on a hollow iron ring and estimating their strength from the deformation or the penetration of the block. For larger detonators of between one-half gram and 1 gram charge as used for borehole shots, the plate would have to be of greater thickness."

3. *Radial lines on lead plates.*—Bigg-Wither, in the article mentioned above, describes in considerable detail tests made with different detonators. He used lead plates 3 mm. thick for detonators Nos. 1 to 3 and lead plates 5 mm. thick for detonators Nos. 4 to 8.

^a Bigg-Wither, H., Notes on detonators: Trans. Inst. Min. Eng., vol. 21, 1900, p. 442.

^b Mumroe, C. E., Lecture on chemistry and explosives, 1888, pp. 22-23.

^c Guttman, Oscar, Manufacture of explosives, vol. 2, 1896, p. 360.

The lead plates were supported on the edges, and the detonators were placed vertically on the centers of the plates. He further states that after the tests the plates may be taken as direct pictorial records of the efficiency of the detonators but that they do not record the report of the explosion, the recording of which is essential; that the detonating effect is not shown so much by the punctures as by the fine radiating marks upon the surface of the plates; that the fine markings show that the force of the explosion smashes the copper tubing to powder, some of which often adheres to the sides of the plates, and that when there are fine radiating lines around the center there are heavier markings outside. The difference in effect is probably due to the upper part of the fulminate not being completely detonated. The results of tests show that detonators may absorb moisture when stored and emphasize the importance of using a detonator of higher power than would be otherwise actually requisite.

It appears, then, that this test is one that might readily be used to distinguish between good and poor or defective detonators regardless of the charge that they contain, and for this purpose the test appears to have considerable merit. However, as an indication of the relative effectiveness of detonators of different grades, that is, containing different weights of charge, it appears to have little value.

4. *Photographs of flashes from electric detonators.*—De Grave^a conceived the idea that the flash or flame of a detonator might vary with the grade of the detonator, and such was the result of tests made by him. He also showed that there was little, if any, difference whether the electric detonator was of high or low tension. The following table gives the results for low-tension detonators:

Results of photographs of flashes of low-tension detonators.

Grade No.—	Dimension of flash.
	<i>Inches.</i>
3	1.0 by 0.22
6	1.6 by .22
7	1.75 by .22
7	1.75 by .22
8	2.0 by .22
8	2.0 by .22

This test was rather unique, but from the results of tests reported it is evident that this test offers no advantage over that of the simple determination of the weight of charge contained within the detonator.

5. *Ability of detonator to explode similar detonators.*—This test is fully stated in a circular dated September 10, 1903, issued by the chief inspector of explosives (Great Britain) to the manufacturers

^a Photographs of flashes of electric detonators: Trans. Inst. Min. Eng., vol. 15, 1897, p. 203.

and importers of detonators. The detonator is there defined^a as "A capsule or case of such strength and construction and containing one or the other of the following explosives of the fulminate class in such quantities that the explosion of one capsule or case will communicate the explosion to other capsules or cases: (1) Fulminate of mercury, (2) fulminate of mercury and chlorate of potash, (3) other compositions."

It is obvious from the definition that with this test no discrimination between the detonators of different grades is possible.

6. *Effect on lead block when detonator is fired in bore hole.*—At the Massachusetts Institute of Technology in 1888–89, tests were conducted by Robert C. Williams and J. B. Seager and reported by Frederick W. Clark.^b

Tests were made of 20 explosives, triple and quintuple detonators (caps) being used. In order that some of the effect of the detonator itself might be eliminated its effect was determined in the following way: The lead block used was a frustum of a cone 5½ inches high, 5½ inches in diameter at the bottom, and 5 inches in diameter at the top. The axial bore was also a frustum of a cone three-fourths of an inch in diameter at the top, five-eighths of an inch in diameter at the bottom, and 2½ inches deep. In casting the blocks the lead was poured when "just barely melted"; the finished block weighed about 45 pounds. The detonator was placed in the bore hole, tamped with dry quartz sand, and fired by means of fuse. As the detonators were slightly less than one-fourth of an inch in diameter the distance between the caps and the walls of the bore hole averaged three-sixteenths of an inch. A tabulation of the results of the tests follows:

Results of firing detonators in bore holes of lead blocks.

Grade of detonator (cap).	Capacity of bore hole.		Difference.	Average.
	Before firing detonator.	After firing detonator.		
	<i>C. c.</i>	<i>C. c.</i>	<i>C. c.</i>	<i>C. c.</i>
"Eagle" triple ^a	14.3	17.0	2.7	2.3
Do.....	14.3	16.3	2.0	
Do.....	14.3	16.6	2.3	
"Eagle" quintuple ^b	14.3	17.2	2.9	3.1
Do.....	14.3	17.5	3.2	

^a At that time the commercial grade name of the Pittsburgh testing station No. 3 detonator.

^b At that time the commercial grade name of the Pittsburgh testing station No. 5 detonator.

It is evident that the method of conducting these tests was such that only a part of the energy of the detonator was represented by the expansion of the bore hole because much of the energy was

^a Practical Coal Mining, vol. 2, 1903, p. 237.

^b Some tests of the relative strength of nitroglycerin and other explosives: Trans. Am. Inst. Min. Eng., vol. 18, 1890, p. 515.

used to disintegrate and pulverize the sand. This was proven by tests made at the Pittsburgh testing station with electric detonators containing similar charges. A No. 3 electric detonator when fired in a cast-lead block with a bore hole of such size that the detonator would fit snugly within it produced an expansion of 5.8 c. c. A similar test with a No. 5 electric detonator gave an expansion of 9.2 c. c.

In the tests at the Massachusetts Institute of Technology, two detonators fired simultaneously within the bore hole produced considerably more than twice the expansion produced by one detonator, probably because the distance between the charge and the sides of the bore hole was less and, accordingly, the charging density was increased. The following tabulated results show this:

Results of firing simultaneously two detonators in bore hole of lead block.

Grade of detonator (cap).	Capacity of bore hole—		Difference.	Average.
	Before firing detonator.	After firing detonator.		
	C. c.	C. c.	C. c.	C. c.
"Eagle" triple.....	14.3	21.9	7.6	6.8
Do.....	14.3	20.4	6.1	
"Eagle" quintuple.....	14.3	20.0	9.7	

Further lead-block tests were made with 13 sensitive explosives, both triple and quintuple detonators (caps) being used. The charge consisted of 6 grams of explosive, loaded and fired as previously described. The conclusion drawn was that explosives when fired with a quintuple detonator produce 9.7 per cent greater expansion than that produced with a triple detonator.

It is evident that in arriving at this conclusion the author did not take into consideration the fact that the quintuple detonator had a charge of 0.80 gram of fulminating composition, that the triple detonator had only a 0.54-gram charge, and that therefore the weight of the total charge, including the quintuple detonator, was increased 4.0 per cent over the weight of the total charge when triple detonators were used. Furthermore, the 4.0 per cent increase in weight represented principally mercury fulminate, a powerful, quick-acting explosive which, under the conditions of the tests, would exert its full effect in enlarging the bore hole. From the data presented, the results can not be properly interpreted as indicating that, with small charges (in this case 6 grams) of an explosive detonating directly under the influence of a detonator, an increase of the force of the explosive was obtained with a detonator of the higher grade.

The results of tests made at the Pittsburgh testing station with sensitive explosives do not substantiate the conclusions drawn. In order to differentiate between grades of electric detonators, it was necessary to use large quantities of insensitive explosives under conditions simulating those of actual blasting operations.

TESTS FOR DETERMINING DIRECTLY THE STRENGTH OF P. T. S. S. ELECTRIC DETONATORS.

CHARACTER OF ELECTRIC DETONATORS TESTED.

Tests for determining directly the strength of electric detonators were made with six grades of P. T. S. S. electric detonators (fig. 1).

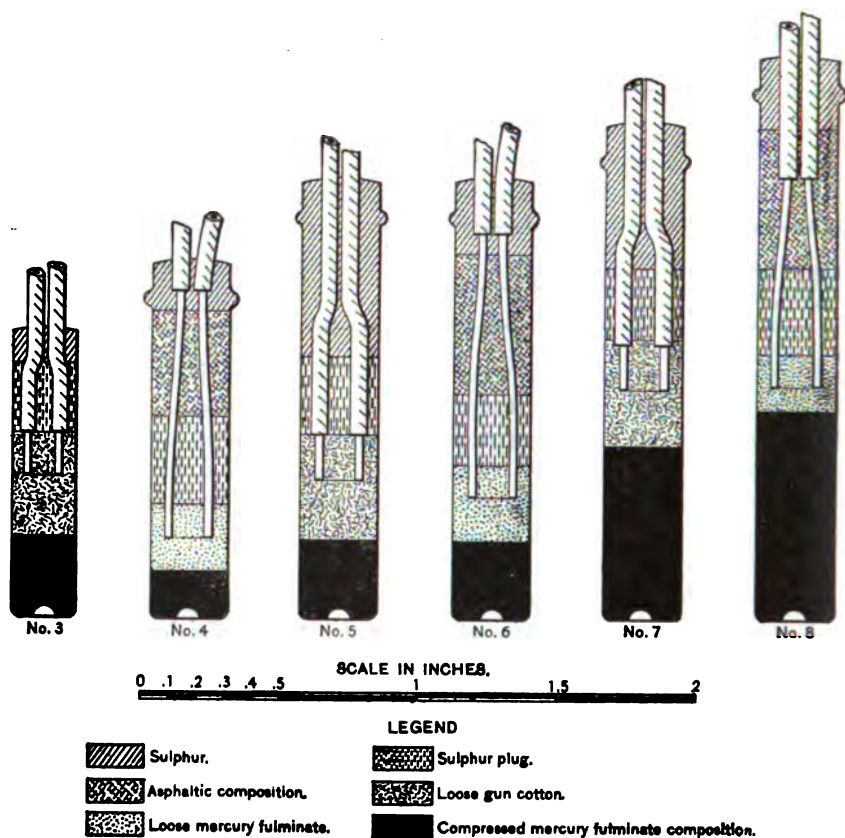


FIGURE 1.—Cross-sectional view of six P. T. S. S. electric detonators.

A physical examination of each showed the results tabulated below. Each measurement represents an average of the measurements of five electric detonators of a given grade.

Results of physical examination of P. T. S. S. electric detonators.

Grade of electric detonator.	Length of shell.	Outside diameter of shell.	Inside diameter of shell.	Thickness of shell.	Length of compressed charge.	Length of priming charge.	Length of sulphur plug.	Length of asphaltic composition, if any.	Length of sulphur filling.
	<i>Inches.</i>	<i>Inches.</i>	<i>Inches.</i>	<i>Inches.</i>	<i>Inches.</i>	<i>Inches.</i>	<i>Inches.</i>	<i>Inches.</i>	<i>Inches.</i>
No. 3.....	1.00	0.234	0.220	0.007	0.28	0.37	0.25		0.10
No. 4.....	1.25	.274	.280	.007	.16	.24	.31	0.38	.16
No. 5.....	1.55	.274	.280	.007	.28	.37	.28		.62
No. 6.....	1.55	.274	.280	.007	.28	.27	.25	.50	.25
No. 7.....	1.75	.274	.280	.007	.62	.38	.25		.50
No. 8.....	2.00	.274	.280	.007	.75	.20	.31	.50	.24

Details of the wiring of the electric detonators tested are given below:

Details of the wiring of six grades of P. T. S. S. electric detonators.

Grade of electric detonator.	Distance wires projected below sulphur plug.	Distance from end of insulation to end of wires.
	<i>Inches.</i>	<i>Inches.</i>
No. 3.....	0.16	0.16
No. 4.....	.12	.88
No. 5.....	.16	.16
No. 6.....	.12	.94
No. 7.....	.19	.16
No. 8.....	.12	.75

The outside diameter and the thickness of the shells were determined with micrometers. The inside diameter of the shells was computed from the figures so determined. For grades Nos. 3, 5, and 7 the priming charge was guncotton. No. 3 electric detonators could not be procured from the manufacturers, so the priming charge, sulphur plug, and sulphur filling were placed in a No. 3 detonator at the Pittsburgh testing station; all other electric detonators were purchased from manufacturers.

The weights and the results of chemical analyses of the charges of the six grades of electric detonators were as follows:

Weights and results of chemical analyses of charges of P. T. S. S. electric detonators.

Grade of electric detonator.	Weight of compressed charge.	Weight of priming charge.	Weight of total charge.	Percentage in compressed charge of—		Percentage in priming charge of—		Percentage in total charge of—		
				Mercury fulminate.	Chlorate of potash.	Guncotton.	Mercury fulminate.	Mercury fulminate.	Chlorate of potash.	Guncotton.
	<i>Grams.</i>	<i>Grams.</i>	<i>Grams.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>
No. 3 <i>a</i>	0.4920	0.0200	0.5120	87.94	12.06	100.00		84.50	11.59	3.91
No. 4 <i>a</i>	.3255	.3230	.6485	88.51	11.49		100.00	94.24	5.76	
No. 5 <i>b</i>	.6990	.0240	.7230	89.13	10.87	100.00		86.17	10.51	3.32
No. 6 <i>b</i>	.6485	.3510	.9995	88.82	11.18		100.00	92.75	7.25	
No. 7 <i>c</i>	1.4854	.0247	1.5101	88.93	11.07	100.00		87.47	10.89	1.64
No. 8 <i>d</i>	1.5110	.3000	1.8110	89.77	10.23		100.00	91.47	8.53	

a Analyst, A. L. Hyde.
b Analyst, W. C. Cope.

c Analyst, C. A. Taylor.
d Analyst, J. H. Hunter.

The results of calorimeter tests are tabulated below:

Results of calorimeter tests of six grades of P. T. S. S. electric detonators.

Grade of electric detonators.	Number of electric detonators used in each test.	Number of tests averaged.	Heat evolved per electric detonator.	Total charge per electric detonator.	Heat evolved per electric detonator on the basis of a charge of 77.7 per cent mercury fulminate and 22.3 per cent chlorate of potash (exact combustion). ^a
			<i>Large calories.</i>	<i>Grams.</i>	<i>Large calories.</i>
No. 3.....	30	1	0.35	0.5120	0.36
No. 4.....	25	2	.48	.6485	.46
No. 5.....	20	2	.49	.7230	.51
No. 6.....	15	2	.62	.9905	.71
No. 7.....	10	2	1.01	1.5101	1.07
No. 8.....	10	2	1.14	1.8110	1.28

^a Berthelot, M., *Explosives and their power*, 1892, p. 470.

The tests were made with the explosives calorimeter^a of the Pittsburgh testing station and the rise in temperature of the water surrounding the bomb was about 0.140° C., an increase too small to insure the most accurate results. Nevertheless, the results are valuable as showing the potential energy of the electric detonators and that the potential energy is approximately a direct function of the total charge. The last column is added to show how close the heat evolved per electric detonator was to that which was to be expected had the mercury-fulminate composition been of mercury fulminate and chlorate of potash in the proportions necessary for exact combustion.

SQUIRTED LEAD BLOCK TESTS.

Tests of the six grades of electric detonators were made with squirted-lead blocks. The blocks were squirted 2 inches in diameter and were cut 3 inches long. The axial bore hole was drilled a depth equal to the length of the electric detonator to be tested and a diameter such that the electric detonator would fit snugly into it. The volume of the bore hole was measured with water before and after firing the shot. The tendency of the squirted blocks, because of their small diameter (2 inches), to bulge around the sides makes a comparison between the low-grade and the high-grade electric detonator more difficult and makes impossible a comparison of the increase in volume with the weight of total charge. Nevertheless, the volume increases with the weight of total charge as is to be expected.

^a Hall, Clarence, Snelling, W. O., and Howell, S. P., *Investigations of explosives used in coal mines*; with a chapter on the natural gas used at Pittsburgh, by G. A. Burrell, and an introduction by C. E. Munroe: Bull. 15, Bureau of Mines, 1912, p. 109.

The results of the tests are tabulated below:

Results of tests P. T. S. S. electric detonators with squirted-lead blocks.

Grade of electric detonator.	Test No.	Volume of bore hole—		Increase of volume after firing electric detonator.	Average increase of volume.	Weight of total charge.
		Before firing detonator.	After firing detonator.			
		<i>C. c.</i>	<i>C. c.</i>	<i>C. c.</i>	<i>C. c.</i>	<i>Grams.</i>
No. 3.....	AA47	0.9	7.5	6.6	6.4	0.5120
	AA48	.9	7.2	6.3		
No. 4.....	AA45	1.35	11.0	9.6	9.5	.6485
	AA46	1.35	10.8	9.4		
No. 5.....	AA20	1.8	13.3	11.5	11.3	.7230
	AA27	1.7	12.6	11.1		
No. 6.....	AA10	1.7	20.0	18.3	18.2	.9995
	AA11	1.7	19.8	18.1		
No. 7.....	AA18	2.1	38.5	36.4	36.7	1.5101
	AA19	1.9	38.9	37.0		
No. 8.....	AA16	2.1	49.7	47.6	47.5	1.8110
	AA17	2.15	49.6	47.45		

* Bottom blown out of block; it was fastened in with paraffin before volume of bore hole was measured.

CAST LEAD BLOCK TESTS.

Tests of the six grades of P. T. S. S. electric detonators were made also with cast-lead blocks. The blocks were cast as solid cylinders 100 mm. in diameter and 100 mm. high. The axial bore hole of each was drilled a depth equal to the length of the electric detonator to be tested, and of a diameter such that the electric detonator would fit snugly into it. The volume of the bore hole was measured with water before and after the shot. When more than two trials were made with any given electric detonator, the two trials that were within 5 per cent variation were selected for averaging. A comparison of the average increase of volume (y) with increase of the weight of total charge (x) shows that the relation $y=15.5$ ($x=0.12$) is closely maintained.

Plate I shows the comparative effects of the different electric detonators on the cast-lead blocks.

The details of the cast lead block tests are tabulated below:

Results of tests with cast-lead blocks of P. T. S. S. electric detonators.

Grade of electric detonator.	Test No.	Volume of bore hole—		Increase of volume.	Average increase of volume.	Weight of total charge.	Increase of volume as compared with total charge, by formula $y=15.5$ ($x=0.12$).
		Before firing electric detonator.	After firing electric detonator.				
		<i>C. c.</i>	<i>C. c.</i>	<i>C. c.</i>	<i>C. c.</i>	<i>Grams.</i>	
No. 3.....	AA49	0.9	6.6	5.7	5.8	0.5120	6.1
	AA50	.9	6.8	5.9			
No. 4.....	AA51	1.35	9.3	7.95	7.8	.6485	8.2
	AA52	1.35	9.1	7.75			
No. 5.....	AA3	1.7	11.1	9.4	9.2	.7230	9.3
	AA39	1.7	10.7	9.0			
No. 6.....	AA30	1.7	16.0	14.3	14.0	.9995	13.6
	AA55	1.6	15.2	13.6			
No. 7.....	AA37	1.9	23.0	21.1	21.0	1.5101	21.5
	AA44	1.9	22.8	20.9			
No. 8.....	AA42	2.1	28.6	26.5	26.2	1.8110	26.2
	AA43	2.1	28.0	25.9			

TESTS BY EXPLOSION OF DETONATING FUSE (CORDEAU DETONANT)^a BY INFLUENCE.

The usual method of firing detonating fuse (cordeau detonant) is to place a detonator on the end of the fuse. Some detonators will explode detonating fuse when not in direct contact with it. Hence, in the expectation that the strength of an electric detonator might be determined by varying the distance between the electric detonator and the detonating fuse, trials with a few electric detonators were made in such a way as to fix for each grade a limiting distance at which no explosion would occur, explosion occurring if the distance were lessened 1 mm.

The detonating fuse was arranged in the four different ways indicated in the following tables:

Results of explosion-by-influence tests in which detonating fuse was placed parallel with electric detonator.

Grade of electric detonator.	Test No.	Trial.	Separating distance.	Result.
No. 6.....	M243	a b c d e f g	Mm. 20 10 5 0 0 0	No explosion. Do. Do. Do. Do. Do. Do.
No. 8.....	M245	h a b c	0 0 0 0	Do. Do. Do. Do.

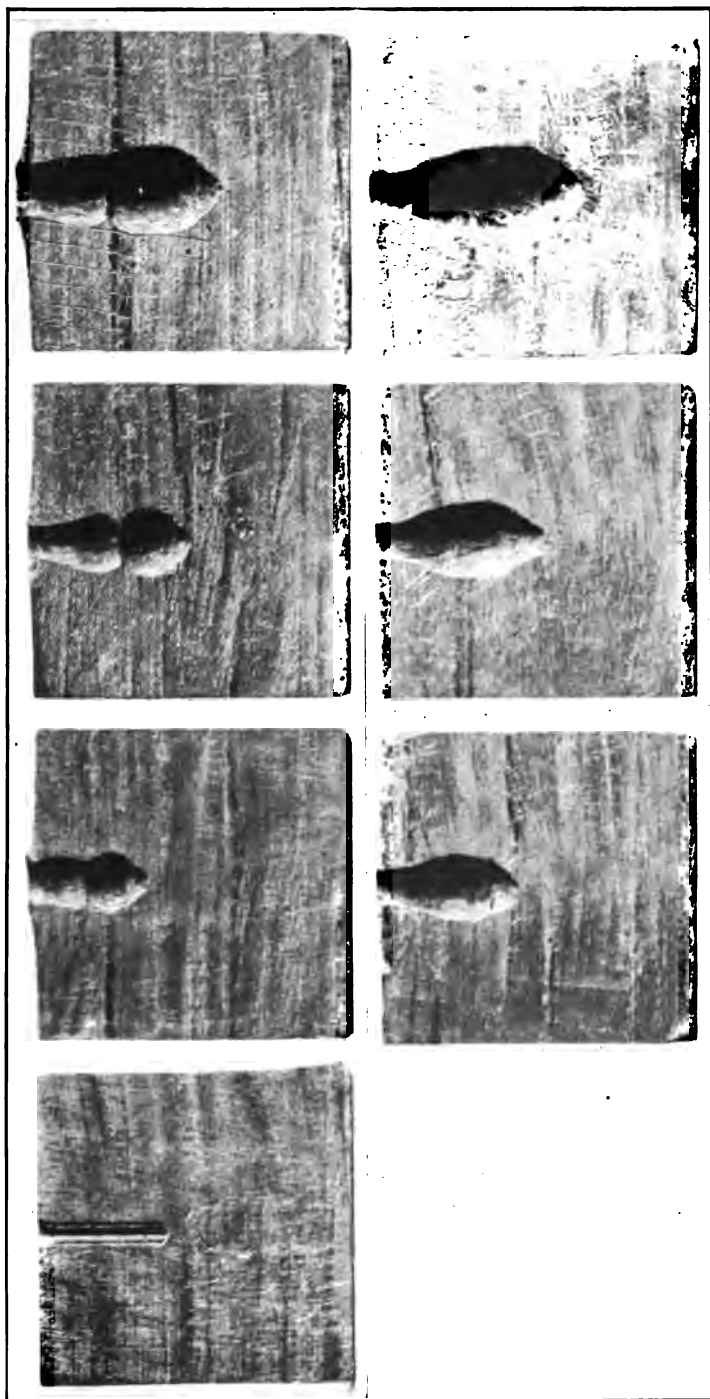
Results of explosion-by-influence tests in which side of detonating fuse touched the end of the electric detonator and was at right angles to it.

Grade of electric detonator.	Test No.	Trial.	Result.
No. 6.....	M244	a b c d	No explosion. Do. Do. Do.
No. 8.....	M246	e a b	Do. Do. Do.

Results of explosion-by-influence tests in which detonating fuse and electric detonator were placed in the same axial line.

Grade of electric detonator.	Test No.	Trial.	Separating distance.	Result.
No. 4.....	M253	a b c d e f g h i j k	Mm. 5 8 6 6 6 7 8 8 8 8	Explosion. No explosion. Do. Do. Explosion. No explosion. Explosion. No explosion. Do. Do. Do.

^a See p. 66.



RESULTS OF CAST LEAD BLOCK TESTS OF P. T. S. ELECTRIC DETONATORS. UPPER LEFT-HAND CORNER. LEAD BLOCK BEFORE TEST; LEFT TO RIGHT, BLOCKS AS AFFECTED BY DETONATORS NOS. 3, 4, 5, 6, 7, AND 8.

Results of explosion-by-influence tests in which detonating fuse and electric detonator were placed in the same axial line—Continued.

Grade of electric detonator.	Test No.	Trial.	Separating distance.	Result.
No. 6.....	M251	a b c d e f g h i j k l m n o	Mm. 3 4 5 6 7 7 7 8 9 10 10 10 10 10	Explosion. Do. Do. Do. No explosion. Do. Explosion. Do. Do. No explosion. Do. Do. Do. Do.
No. 8.....	M247	a b c d e f g h i j k l	0 0 10 5 1 3 4 4 5 5 5 5	Explosion. Do. No explosion. Do. Explosion. Do. No explosion. Explosion. No explosion. Do. Do. Do.

Results of explosion-by-influence tests in which detonating fuses were placed at right angles to electric detonators but at different distances from them in such a way that axial line of detonating fuse intersected side of electric detonator.

Grade of electric detonator.	Test No.	Trial.	Distance from center line of detonating fuse to end of detonator.	Separating distance.	Result.
No. 4.....	M254	a b c d e f	Mm. 5	Mm. 2 1 2 2 2 2	No explosion. Explosion. No explosion. Do. Do. Do.
No. 6.....	M252	a b c d e f g h i j k l m	7	5 4 3 2 1 0 0 2 2 3 3 3 3	Do. Do. Do. Do. Do. Do. Explosion. No explosion. Explosion. No explosion. Do. Do. Do.
No. 8.....	M250	a b c d e f g h i j	10	0 4 5 5 6 6 6 6 6 6	Explosion. Do. No explosion. Do. Explosion. No explosion. Do. Do. Do. Do.

The above results are so much at variance with the established explosive efficiency of detonators (see pp. 45 and 46) that this method of determining the strength of detonators is considered of little value.

The tests made with the No. 4 and the No. 6 electric detonators placed in the same axial line as that of the detonating fuse would indicate that in actual blasting there may be some advantage gained from inserting the electric detonator in the top of the primer or cartridge. Although it has been impossible to show by tests any loss in energy resulting from the detonation of an explosive when the electric detonator is placed in the side of a primer—that is, having the end of the electric detonator intersect the axial line of the primer, it is believed that the former method of insertion is preferable. When the top of the primer is opened and an electric detonator is pushed into it and the paper ends of the cartridge are gathered together and bound with twine, the electric detonator is held firmly in place. When this method is used there is less danger of the wires becoming short-circuited, and it is impossible for the end of the detonator to project through the side of the cartridge, a position that would not only tend to reduce its effectiveness, but would also be a source of danger in loading and tamping the drill hole.

TESTS BY DEPRESSION OF LEAD PLATES.

The strength of the electric detonators was also determined directly by tests involving the depression of lead plates. The details of the tests are indicated in the tabulations following:

Results of depression tests of electric detonators placed on end on $\frac{1}{4}$ -inch lead plates.^a

Grade of electric detonator.	Test No.	Volume of water held in depression. ^b	Diameter of crater.	Depth of crater.	Height of cone on bottom.
		<i>C. c.</i>	<i>Mm.</i>	<i>Mm.</i>	<i>Mm.</i>
No. 3.....	M264	0.35	11	7	2
No. 4.....	M262	.50	13	7	4
No. 5.....	M167	.40	13	6	4
No. 6.....	M149	.40	13	7	3
No. 7.....	M151	.40	13	6	2
No. 8.....	M147	.35	13	7	2

^a See Pl. II.

^b The measurement of volume, which was determined with water, was unsatisfactory because of the action of surface tension, and the results are accurate only within 50 per cent. No result obtained agrees even approximately with the established results of the explosive efficiency of electric detonators (see p. 45).

Results of depression tests of electric detonators placed on side on $\frac{1}{4}$ -inch lead plates.^a

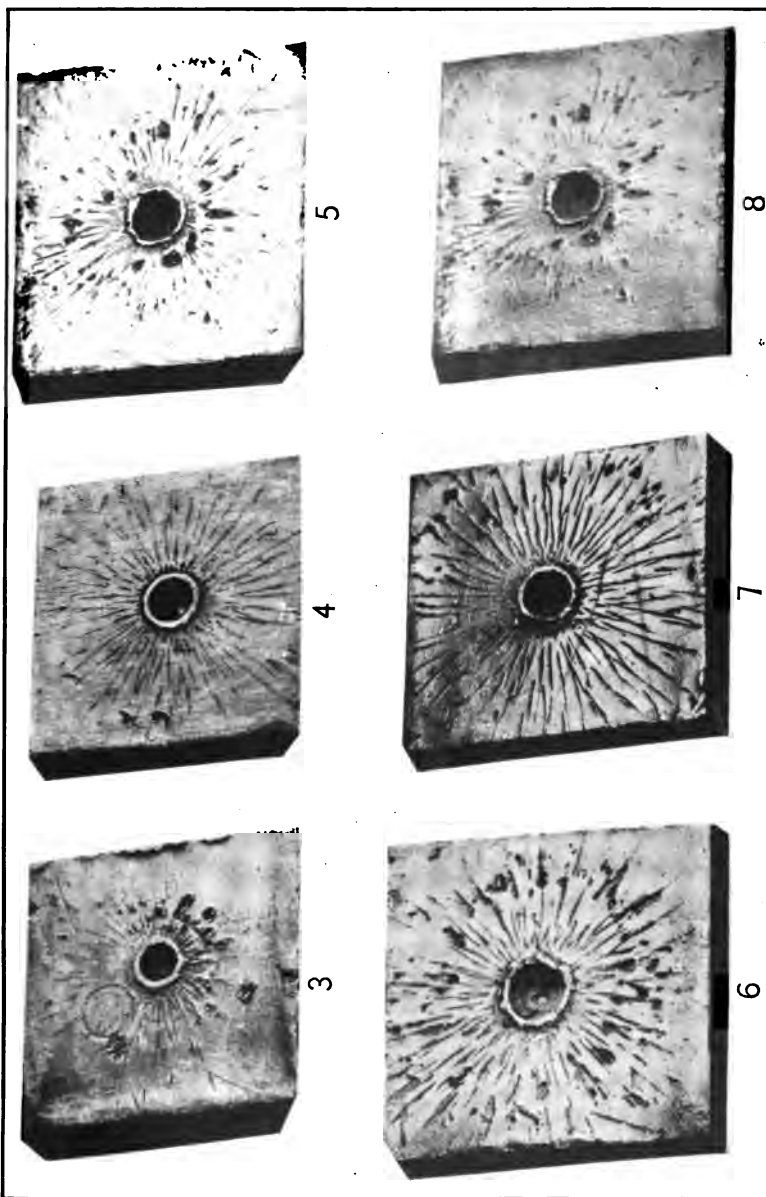
MEASURED WITH WATER.

Grade of electric detonator.	Test No.	Volume. ^b	Length of crater.	Width of crater.	Depth of crater.
		<i>C. c.</i>	<i>Mm.</i>	<i>Mm.</i>	<i>Mm.</i>
No. 3.....	M265	0.30	10	10	4
No. 4.....	M263	.25	15	10	4
No. 5.....	M158	.40	15	11	4
No. 6.....	^c M150	.50	21	13	5
No. 7.....	^c M152	.60	20	14	5
No. 8.....	M148	.90	31	14	6

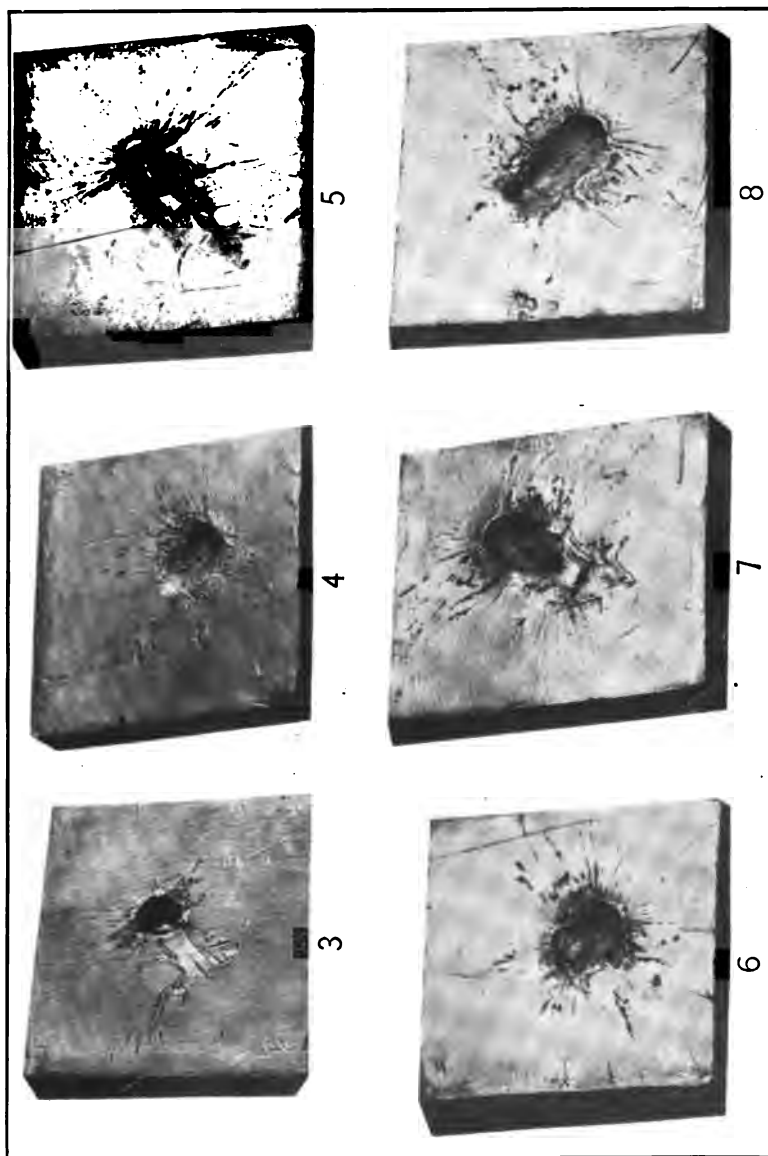
^a See Pl. III.

^b See footnote of preceding table.

^c Bottom of plate slightly raised; not raised in other tests.



SCORING OF LEAD PLATES BY P. T. S. ELECTRIC DETONATORS NOS. 3, 4, 5, 6, 7, AND 8 PLACED ON END.



SCORING OF LEAD BLOCKS BY P. T. S. ELECTRIC DETONATORS NOS. 3, 4, 5, 6, 7, AND 8 LAID ON SIDE.

Results of depression tests of electric detonators placed on side on $\frac{1}{4}$ -inch lead plates—Con.

MEASURED WITH SAND.^a

Grade of electric detonator.	Test No.	Plate No.	Weight of sand contained in depression No.—					Average of five measurements.	Grand average.	Volume. ^b
			1	2	3	4	5			
			Grams.	Grams.	Grams.	Grams.	Grams.	Grams.	Grams.	C. c.
No. 3.....	M302	1	0.129	0.122	0.121	0.142	0.146	0.132		
		2	.191	.191	.194	.172	.197	.189		
No. 4.....	M303	3	.226	.209	.235	.221	.229	.224	0.182	0.128
		1	.379	.405	.376	.405	.365	.386		
No. 5.....	M304	2	.327	.325	.333	.302	.312	.320		
		3	.415	.393	.397	.404	.386	.399	.368	.259
No. 6.....	M301	1	.361	.359	.340	.355	.381	.359		
		2	.382	.380	.393	.379	.390	.385		
No. 7.....	M306	3	.361	.365	.377	.355	.366	.365	.370	.261
		1	.574	.565	.620	.620	.590	.594		
No. 8.....	M306	2	.580	.586	.607	.599	.586	.588		
		3	.622	.600	.615	.601	.598	.607	.596	.420
No. 9.....	M306	1	.891	.867	.890	.890	.892	.884		
		2	1.002	.994	1.034	.994	1.002	1.005		
No. 10.....	M306	3	1.114	1.076	1.081	1.108	1.144	1.105	.998	.703
		1	1.183	1.173	1.230	1.150	1.244	1.196		
No. 11.....	M306	2	1.245	1.237	1.252	1.272	1.216	1.244		
		3	1.269	1.252	1.257	1.280	1.320	1.272	1.237	.871

^a The sand was fine and dry.

^b The volume was computed from the grand average weight by dividing it by the specific gravity of the sand, which was 1.42. This test was acceptable because the volume of the depression varied approximately as the explosive efficiency of the electric detonator.

THE NAIL TEST.

It was evident that the methods previously used for the direct determination of the relative strength of detonators were not satisfactory or accurate. During the latter part of the investigation an endeavor was made to devise a test that would give results approximating those obtained by the indirect tests.

In the tests made with the four No. 6 detonators having different compositions, described later, each electric detonator caused the same amount of energy to be developed from both sensitive and insensitive explosives. However, by the direct methods of testing detonators, one of the No. 6 detonators showed a much higher calorific value than any of the others, and one developed a much greater enlargement of the lead block. Nevertheless it was concluded that although the temperature developed and the volume of gases produced are functions of the efficiency of detonators, the rate of detonation or the rapidity with which the gases are developed is the prime factor and any tests that emphasized this factor should be given consideration.

The test finally decided upon is known as the nail test. This test depends on the angle formed by a nail when a detonator or electric detonator is fired in close proximity to it. For simplicity and cheapness the nail test commends itself.

Four-inch wire finishing nails (20-d.) are used in the test. For the tests herein reported the nails were selected so that they were approx-

imately of the same length, the same gage, and the same weight. The bottom of the electric detonator was placed $1\frac{1}{2}$ inches from the face of the head of the nail and was laid parallel to the nail and separated from it by two 22-gage (0.025-inch) copper wires that were wrapped around the electric detonator. The electric detonator was fastened in position by one strand of a similar copper wire, which was wrapped around it and the nail midway between the ends of the electric detonator. The whole was suspended horizontally in the air in such a manner

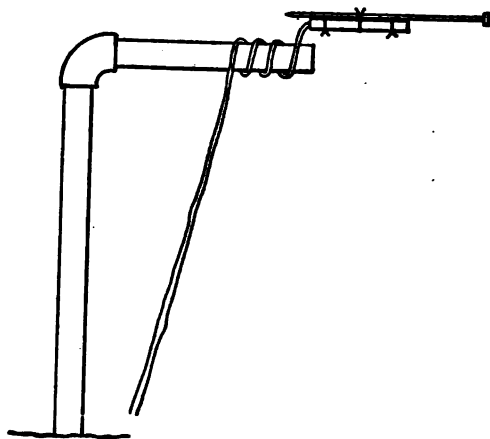


FIGURE 2.—Nail in position for test of electric detonator.

that the nail was directly above the electric detonator, which was then fired. (Fig. 2.) The impact of the exploding electric detonator bent the nail and projected it upward. Care was taken that the nail was not hurled against any solid surface and further distorted.

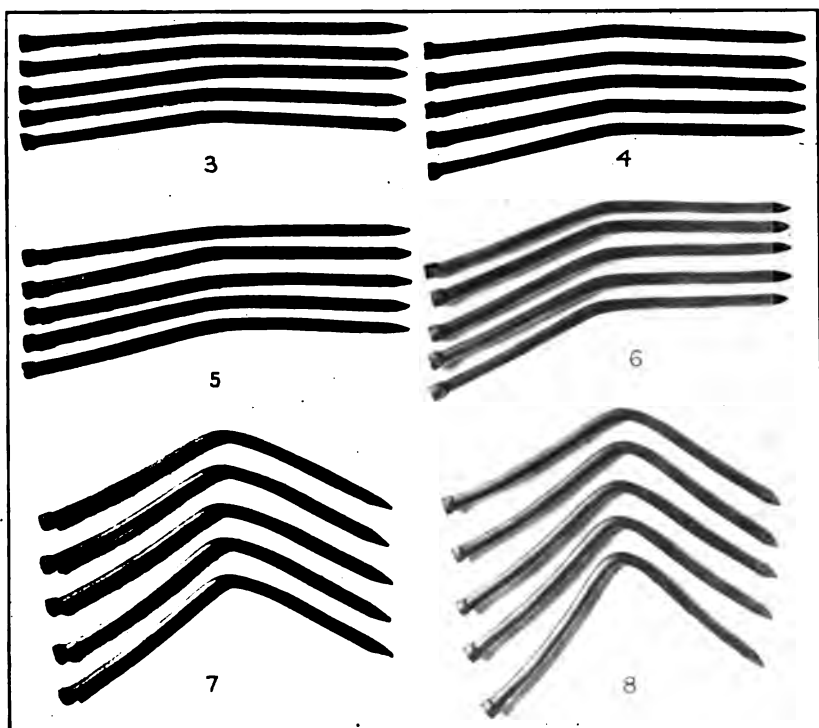
Five trials were made with each grade of electric detonator. The angle through which the nail was bent from its normal position was measured. The angle (average of five trials) was taken as a measure of the strength of the electric detonator. The results were as follows:

Results of nail tests ^a of six grades of P. T. S. S. electric detonators.

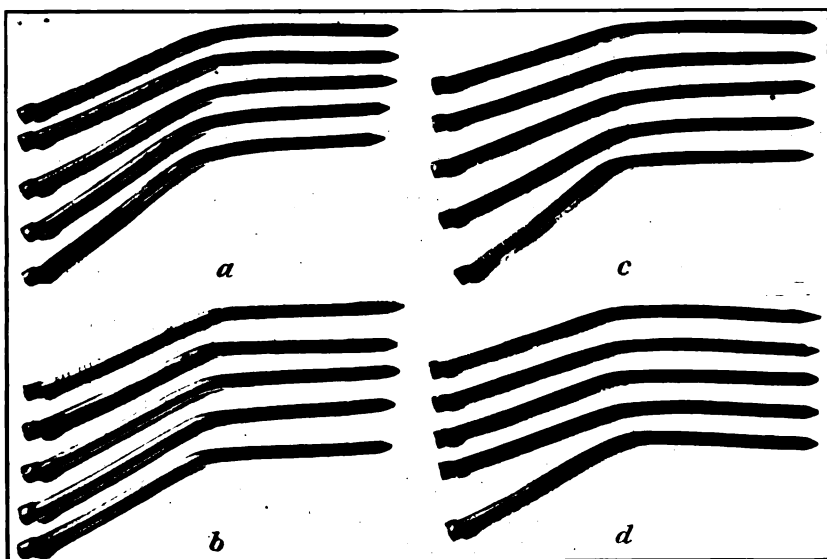
Grade of electric detonator.	Test No.	Angle of bending resulting from trial No. —					Average.
		1	2	3	4	5	
No. 3.....	M279	12	10	8	9	7	9.2
No. 4.....	M280	12	11	14	11	16	12.8
No. 5.....	M281	11	13	14	13	8	11.8
No. 6.....	M288	23	24	25	24	26	24.4
No. 7.....	M283	60	50	53	54	59	55.2
No. 8.....	M284	68	78	76	86	98	81.2

^a See Pl. IV, A.

The variation in the results of individual trials was largely due to variation in the individual electric detonators. An attempt was made to get more uniform results with annealed nails, but with these there was practically the same variation in results. In such tests, as well as in all physical tests of explosives, discrepancies resulting from unavoidable sources of error can not be eliminated, and, accordingly, only averages should be considered in comparing the practical value of the electric detonators.



A. RESULTS OF NAIL TESTS OF P. T. S. S. ELECTRIC DETONATORS NOS. 3, 4, 5, 6, 7, AND 8.



B. RESULTS OF NAIL TESTS OF NO. 6 ELECTRIC DETONATORS. *a*, WESTERN COAST; *b*, SPECIAL; *c*, P. T. S. S.; *d*, FOREIGN.

TESTS FOR DETERMINING INDIRECTLY THE STRENGTH OF P. T. S. S. ELECTRIC DETONATORS.

Details of different forms of tests made to determine indirectly the strength of P. T. S. S. electric detonators are given below.

RATE-OF-DETONATION TESTS.*

The rate of detonation of the explosive was determined by placing the cartridges end to end in a 28-gage (B. & S.) galvanized-iron tube 42 inches long, which was of slightly larger diameter than the cartridges. The paper ends of each cartridge were cut off squarely in order that the explosive material of the cartridges would be continuous throughout the file, which was a little more than 1 meter long. Four copper wires were inserted through perforations in the tube and the cartridge file so that the distance between adjacent wires made it possible to determine the rate of detonation through the first quarter meter, the second quarter meter, the last half meter, and the entire meter, and the data were so recorded.

Each wire carried an electric current and was attached to a Metteng recorder in such a way that at the instant the wire was broken a spark was recorded on a rapidly moving soot-covered drum. From the sparks thus recorded and the speed of the drum, the time interval between the breaking of the wires in the meter file was computed and was expressed as rate of detonation in meters per second.

The rate-of-detonation tests were carried on with different explosives as described below.

TESTS WITH AN EXPLOSIVE OF CLASS 1, SUBCLASS a.

In one series of tests sample 1 of an explosive of class 1, subclass a—an ammonium-nitrate explosive containing a sensitizer that is itself an explosive—was used. The cartridges were seven-eighths of an inch in diameter. The results were as follows:

Results of rate-of-detonation tests with sample 1 of an explosive of class 1, subclass a.

Grade of electric detonator.	Test No.	Rate of detonation in tube.							
		First quarter.		Second quarter.		Second half.		Full length.	
		Individual rate.	Average rate.	Individual rate.	Average rate.	Individual rate.	Average rate.	Individual rate.	Average rate.
		<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>
No. 3.....	M242	Detonation complete; 16 inches of explosive used.							
	M242	3 inches blown off; 16 inches of explosive used.							
	M248	Detonation complete; 16 inches of explosive used.							
No. 4.....	M248	Detonation complete; 16 inches of explosive used.							
	M248	Detonation complete; 16 inches of explosive used.							
No. 5.....	D963	2, 921		1, 956 20 inches detonated.					
	D960	2 inches blown off.							
No. 6.....	D966	2, 445	2, 458	2, 678	2, 632	2, 205	2, 205	2, 368	2, 362
	D967	2, 472		2, 586		2, 205		2, 356	
No. 7.....	D964	2, 777	2, 850	2, 586	2, 571	2, 331	2, 374	2, 521	2, 528
	D965	2, 922		2, 556		2, 368		2, 535	
No. 8.....	D968	2, 184	2, 217	2, 250	2, 285	2, 472	2, 426	2, 337	2, 334
	D969	2, 250		2, 320		2, 380		2, 331	
Grand average.		2, 508		2, 496		2, 335		2, 408	

* For more detailed description of this test, see Bull. 15, Bureau of Mines: Investigations of explosives used in coal mines; with a chapter on the natural gas used at Pittsburgh, by G. A. Burrell, and an introduction by C. E. Munroe, by Clarence Hall, W. O. Snelling, and S. P. Howell, 1912, pp. 92-96.

No detonation occurred in those tests in which 2 or 3 inches of the cartridge was blown off. In those tests in which 16 inches of the explosive was used no attempt was made to determine the rate of detonation.

The grand average indicates that the rate fell off in the last half meter. Individual tests with a given detonator showed remarkable uniformity for each electric detonator of Nos. 6, 7, and 8, and with No. 6 the maximum rate was obtained in the second quarter, with No. 7 in the first quarter, and with No. 8 in the second half. The average rates for the full length of the tube did not vary greatly.

The percentage of complete detonations with each detonator was as follows:

Percentage of complete detonations in rate-of-detonation tests with sample 1 of an explosive of class 1, subclass a.

Grade of electric detonator.	Number of tests.	Number of tests in which incomplete detonation occurred.	Complete detonations.
			<i>Per cent.</i>
No. 3.....	3	1	67
No. 4.....	2	0	100
No. 5.....	2	1	50
No. 6.....	2	0	100
No. 7.....	2	0	100
No. 8.....	2	0	100

These tests show that an explosive of class 1, subclass *a*, that is insensitive, tends more readily to become completely detonated with the higher grades of electric detonators, but that if the explosive detonates at all its rate is independent of the grade of electric detonator used.

The results of tests with sample 2 of an explosive of class 1, subclass *a*, follow. The cartridges used were 1½ inches in diameter.

Results of rate-of-detonation tests with sample 2 of an explosive of class 1, subclass a.

Grade of electric detonator.	Test No.	Rate of detonation in tube.							
		First quarter.		Second quarter.		Second half.		Full length.	
		Individual rate.	Average rate.	Individual rate.	Average rate.	Individual rate.	Average rate.	Individual rate.	Average rate.
		<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>
No. 3.....	D1154	3,090	2,977	3,423	5,529	4,784	3,064	3,854	2,668
	D1155	2,973		5,946		2,973		3,308	
	D1156			5,706		3,708			
	D1157	2,884		7,287		3,435		3,780	
No. 7.....	D1158	2,980	3,324	5,498	4,698	3,516	3,340	3,673	3,506
	D1149	3,836		4,198		3,089		3,477	
	D1150	2,973		6,286		2,935		3,398	
	D1181	2,649		5,174		3,739		2,615	
	D1182	3,836		3,184		3,618		2,582	

* Test No. D1156 not included in average.

The averages for each detonator show a positive acceleration in the second quarter meter and a negative acceleration in the second half meter. The rates for the meter length is within the experimental error, and they were, therefore, practically uniform.

The results of tests with sample 3 of an explosive of class 1, subclass *a*, follow. The cartridges used were $1\frac{1}{2}$ inches in diameter.

Results of rate-of-detonation tests with sample 3 of an explosive of class 1, subclass a.

Grade of electric detonator.	Test No.	Rate of detonation in tube.							
		First quarter.		Second quarter.		Second half.		Full length.	
		Individual rate.	Average rate.	Individual rate.	Average rate.	Individual rate.	Average rate.	Individual rate.	Average rate.
No. 3.....	D1161	Meters per second. 3, 143		Meters per second. 3, 571		Meters per second. 2, 980		Meters per second. 3, 488	
	D1162	4, 167		3, 134		3, 397		3, 532	
	D1163	4, 450	4, 308	3, 352	3, 352	3, 188	3, 188	3, 510	3, 510
No. 7.....	D1127	3, 477		4, 045		3, 450		3, 589	
	D1128	3, 007		4, 363		3, 202		3, 371	
	D1129	3, 090		4, 541		3, 296		3, 477	
	D1130	4, 231	3, 451	3, 729	4, 170	3, 359	3, 327	3, 636	3, 518

^a Detonation incomplete; test not averaged.

The average rate for the meter length was practically uniform.

The percentage of complete detonations for each electric detonator was as follows:

Percentage of complete detonations in rate-of-detonation tests with sample 3 of an explosive of class 1, subclass a.

Grade of electric detonator.	Number of tests.	Number of tests in which complete detonation occurred.	Complete detonations.
No. 3.....	3	2	Per cent. 67
No. 7.....	4	4	100

TESTS WITH AN EXPLOSIVE OF CLASS 1, SUBCLASS *b*

The results of tests with sample 1 of an explosive of class 1, subclass *b*—an ammonium-nitrate explosive containing a sensitizer that is not in itself an explosive—follow. The diameter of the cartridges used was $1\frac{1}{2}$ inches.

Results of rate-of-detonation tests with sample 1 of an explosive of class 1, subclass b.

Grade of electric detonator.	Test No.	Remarks.
No. 3.....	M237	No detonation.
No. 4.....	M237	Do.
No. 5.....	D870	Do.
No. 6.....	D871	Do.
	D869	Do.
	D871	Do.
No. 7.....	D869	Do.
	D871	Do.
No. 8.....	D869	Do.
	D871	Do.

These tests failed to make a discrimination between the different grades of electric detonators, and hence for the purposes of this investigation were useless.

The results of tests with sample 2 of an explosive of class 1, subclass *b*, follow. The cartridges used were 1½ inches in diameter.

Results of rate-of-detonation tests with sample 2 of an explosive of class 1, subclass b.

Grade of electric detonator.	Test No.	Rate of detonation in tube.							
		First quarter.		Second quarter.		Second half.		Full length.	
		Individual rate.	Average rate.	Individual rate.	Average rate.	Individual rate.	Average rate.	Individual rate.	Average rate.
No. 3.....	M233	6 inches of cartridge blown off.		6 inches of cartridge blown off.		6 inches of cartridge blown off.		6 inches of cartridge blown off.	
	M241	4 inches of cartridge blown off.		4 inches of cartridge blown off.		4 inches of cartridge blown off.		4 inches of cartridge blown off.	
	M241	4 inches of cartridge blown off.		4 inches of cartridge blown off.		4 inches of cartridge blown off.		4 inches of cartridge blown off.	
	M240	3 inches of cartridge blown off.		3 inches of cartridge blown off.		3 inches of cartridge blown off.		3 inches of cartridge blown off.	
	M240	3 inches of cartridge blown off.		3 inches of cartridge blown off.		3 inches of cartridge blown off.		3 inches of cartridge blown off.	
No. 4.....	M231	6 inches of cartridge blown off.		6 inches of cartridge blown off.		6 inches of cartridge blown off.		6 inches of cartridge blown off.	
	M241	4 inches of cartridge blown off.		4 inches of cartridge blown off.		4 inches of cartridge blown off.		4 inches of cartridge blown off.	
	M241	4 inches of cartridge blown off.		4 inches of cartridge blown off.		4 inches of cartridge blown off.		4 inches of cartridge blown off.	
	M240	4 inches of cartridge blown off.		4 inches of cartridge blown off.		4 inches of cartridge blown off.		4 inches of cartridge blown off.	
	M240	4 inches of cartridge blown off.		4 inches of cartridge blown off.		4 inches of cartridge blown off.		4 inches of cartridge blown off.	
No. 5.....	M240	4 inches of cartridge blown off.		4 inches of cartridge blown off.		4 inches of cartridge blown off.		4 inches of cartridge blown off.	
	M241	4 inches of cartridge blown off.		4 inches of cartridge blown off.		4 inches of cartridge blown off.		4 inches of cartridge blown off.	
	M241	4 inches of cartridge blown off.		4 inches of cartridge blown off.		4 inches of cartridge blown off.		4 inches of cartridge blown off.	
	D923	4 inches of cartridge blown off.		4 inches of cartridge blown off.		4 inches of cartridge blown off.		4 inches of cartridge blown off.	
	D926	4 inches of cartridge blown off.		4 inches of cartridge blown off.		4 inches of cartridge blown off.		4 inches of cartridge blown off.	
No. 6.....	M240	Detonation complete; 16 inches of explosive used.							
	D930	8 inches of cartridge blown off.							
	D931	2,000	2,000	3,750	3,750	3,333	3,333	3,333	3,333
	D932	8 inches of cartridge blown off.							
	D1110	5 inches of cartridge blown off.							
No. 7.....	M241	Detonation complete; 16 inches of explosive used.							
	M241	Detonation complete; 16 inches of explosive used.							
	D928	3,358	3,384	3,462	3,589	3,743	3,430	3,475	3,415
	D929	3,462		3,814		3,082		3,333	
	D956	3,333		3,491		3,464		3,437	
No. 8.....	M241	Detonation complete; 16 inches of explosive used.							
	D925	2,647	3,072	3,214	3,290	4,286	3,544	3,462	3,319
	D927	3,169		3,750		3,147		3,285	
	D954	3,235		3,085		3,358		3,247	
	D955	3,255		3,142		3,584		3,283	
Grand average.		3,180		3,460		3,475		3,357	

It is probable that no detonation occurred in those tests in which 3 to 8 inches of the cartridge was blown off. In the trial listed under test M 241 only 16 inches of explosive was used and no attempt was made to determine the rate of detonation.

The grand average shows the tendency of the rate of detonation to increase beyond the first quarter.

It is interesting to observe that the rate of detonation for that 10 centimeters of a 1½-inch cartridge just beyond the electric detonator, as determined with the cordeau detonant, was as follows: For a No. 7 electric detonator, 3,387 meters per second; for a No. 8 electric detonator, 3,387 meters per second.

The percentage of complete detonations for each detonator was as follows:

Percentage of complete detonations in rate-of-detonation tests with sample 2 of an explosive of class 1, subclass b.

Grade of electric detonator.	Number of tests.	Number of tests in which incomplete detonation occurred.	Complete detonations.
No. 3.....	5	5	Per cent. 0
No. 4.....	5	5	0
No. 5.....	5	5	0
No. 6.....	5	5	40
No. 7.....	5	0	100
No. 8.....	5	0	100

These tests show that an explosive of class 1, subclass *b*, that is insensitive, tends more readily to become completely detonated with the higher grades of electric detonators, but that if the explosive detonates at all its rate is independent of the grade of electric detonator used.

TESTS WITH A 20 PER CENT "STRAIGHT" NITROGLYCERIN DYNAMITE.

The results of tests with a 20 per cent "straight" nitroglycerin dynamite follow. The cartridges were seven-eighths of an inch in diameter.

Results of rate-of-detonation tests with a 20 per cent "straight" nitroglycerin dynamite.

Grade of electric detonator.	Test No.	Rate of detonation in tube.							
		First quarter.		Second quarter.		Second half.		Full length.	
		Individual rate.	Average rate.	Individual rate.	Average rate.	Individual rate.	Average rate.	Individual rate.	Average rate.
		<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>
No. 3.....	{ D1096 D1097	2,528 2,781	2,654	{ 3,648 2,967	3,308	{ 2,853 3,156	3,004	{ 2,918 3,007	2,962
No. 4.....	{ D1098 D1099	2,418 2,781	2,600	{ 3,423 2,967	3,196	{ 2,834 2,871	2,852	{ 2,834 2,871	2,852
No. 5.....	{ D992 D993	3,225 2,747	2,986	{ 2,781 2,928	2,854	{ 2,908 2,987	2,948	{ 2,947 2,908	2,928
No. 6.....	{ D1000 D1001 D1002	3,729 3,034 3,947	3,838	{ 2,683 3,034 2,443	2,563	{ 2,767 3,121 3,309	3,038	{ 2,933 3,077 3,158	3,046
No. 7.....	{ D1003 D1004	3,836 3,125	3,480	{ 2,500 2,586	2,543	{ 2,947 3,192	3,070	{ 2,986 3,000	2,993
No. 8.....	{ D1005 D1006	3,358 3,261	3,310	{ 2,679 2,778	2,728	{ 2,980 3,041	3,010	{ 2,980 3,020	3,000
Grand average.			3,145		2,865		2,987		2,964

^a Average rate for the first half meter; rate not included in average.

In the tests where electric detonators Nos. 3, 4, and 5 were used a considerably lower rate of detonation occurred in the first quarter than in the tests where electric detonators Nos. 6, 7, and 8 were used.

Detonation was complete with every grade of electric detonator.

The figures in the grand average indicate that the rate was influenced by a negative acceleration in the second quarter meter, followed by a positive acceleration in the second half meter, though the contrary was true for electric detonators of grades Nos. 3 and 4. All tests except test D993 conformed to this.

The uniformity of the rates for the last half meter and for the meter for every grade are noteworthy; this uniformity held for individual tests as well as for averages.

TESTS WITH A 40 PER CENT STRENGTH AMMONIA DYNAMITE CONTAINING NITROSUBSTITUTION-COMPOUNDS.

The results of tests with a 40 per cent strength ammonia dynamite containing nitrosubstitution compounds follow. The cartridges used were seven-eighths of an inch in diameter and had been repacked.

Results of rate-of-detonation tests with a 40 per cent strength ammonia dynamite containing nitrosubstitution compounds.

Grade of electric detonator.	Test No.	Rate of detonation in tube.							
		First quarter.		Second quarter.		Second half.		Full length.	
		Individual rate.	Average rate.	Individual rate.	Average rate.	Individual rate.	Average rate.	Individual rate.	Average rate.
		Meters per second.	Meters per second.	Meters per second.	Meters per second.	Meters per second.	Meters per second.	Meters per second.	Meters per second.
No. 5.....	• D878	2,136	2,498	3,929	3,014	2,045	2,286	2,811	2,412
	D882	2,136		2,586		2,284		2,350	
	D903	2,679		2,538		2,528		2,446	
	D921	2,679		2,538		2,528		2,439	
No. 6.....	• D875	3,666	2,363	2,206	2,594	2,435	2,593	2,659	2,527
	D904	2,368		2,206		2,572		2,446	
	• D905	2,285		3,041		2,813		2,521	
	D919	2,472		2,446		2,663		2,065	
No. 7.....	D920	2,250	2,543	2,446	2,709	2,543	2,619	2,439	2,608
	• D873	2,631		2,273		2,668		2,658	
	D906	2,394		2,273		2,830		2,557	
	• D907	2,367		2,961		2,967		2,641	
No. 8.....	D918	2,616	2,310	2,894	3,146	2,528	2,755	2,647	2,652
	D950	2,619		2,894		2,500		2,619	
	• D872	2,716		3,667		2,453		2,514	
	D881	2,558		3,209		2,280		2,597	
	D910	1,542		2,815		3,345		2,561	
	D911	2,123		3,041		2,711		2,557	
	• D915	2,679		2,744		2,744		2,195	
	D916	2,647		3,000		2,695		2,795	
	D917	2,647						2,752	

• Rate of detonation not averaged.

• Rate for last three-fourths of a meter.

• Rate for first one-half of a meter.

No tests were made with electric detonators Nos. 3 and 4. The average rate for the meter length increased slightly with the grade of electric detonator used. The fastest rate is recorded for the second quarter meter. The figures for the average rates and for most of the individual tests indicate that the rate increased up to a maximum, and then decreased. With some electric detonators the maximum was reached in the first quarter meter, as in test D903; with others in the second quarter meter, as in test D919; and with others in the second half meter, as in test D910.

If it be assumed that the recorded rate was slightly erratic, but had a general tendency to increase to a maximum, and then to decrease toward an asymptotic normal rate, then the results of all the tests conformed to this assumption.

TESTS WITH A 40 PER CENT STRENGTH AMMONIA DYNAMITE.

The results of tests with a 40 per cent strength ammonia dynamite follow. The cartridges used were $1\frac{1}{4}$ inches in diameter.

Results of rate-of-detonation tests with a 40 per cent strength ammonia dynamite.

Grade of electric detonator.	Test No.	Rate of detonation in tube.							
		First quarter.		Second quarter.		Second half.		Full length.	
		Individual rate.	Average rate.	Individual rate.	Average rate.	Individual rate.	Average rate.	Individual rate.	Average rate.
		<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>
No. 3.....	{ D1189	4,018	3,954	{ 4,412	4,532	{ 4,546	4,242	{ 4,369	4,223
	{ D1140	4,327		{ 4,592		{ 4,054		{ 4,245	
	{ D1141	3,516		{ 4,592		{ 4,128		{ 4,064	
No. 8.....	{ D1126	3,437	3,479	{ 4,314	4,868	{ 4,889	4,429	{ 4,293	4,211
	{ D1137	3,250		{ 5,291		{ 4,417		{ 4,213	
	{ D1138	3,760		{ 5,000		{ 3,982		{ 4,128	

Only the No. 3 and the No. 8 electric detonators were used. The average rate for the meter length is practically the same for the two detonators. The rate increased to a maximum in the second half meter and then decreased as shown by averages; the results of individual tests confirm this conclusion. The rate in the last half meter corresponded closely with the average rate for the meter length.

TESTS WITH A 35 PER CENT STRENGTH GELATIN DYNAMITE 2 YEARS OLD.

The results of tests with a 35 per cent strength gelatin dynamite (2 years old) follow. The cartridges used were $1\frac{1}{4}$ inches in diameter.

34 INVESTIGATIONS OF DETONATORS AND ELECTRIC DETONATORS.

Results of rate-of-detonation tests with a 35 per cent strength gelatin dynamite (2 years old).

Grade of electric detonator.	Test No.	Length of files blown off or deto- nated.	Percentage inches that detonated.	Average percent- age that deto- nated.	Remarks.
		<i>Inches.</i>	<i>Per cent.</i>	<i>Per cent.</i>	
No. 3.....	M231	6.5	15	15.0	Partial detonation.
No. 4.....	M234	7.5	18	18.5	Do.
No. 5.....	D887	7.0	0		No detonation.
	D888	7.0	0		Do.
	D942	13.0	31		Partial detonation.
	D943	13.0	31	15.5	Do.
No. 6.....	D889	7.0	0		No detonation.
	D890	18.0	43		Partial detonation.
	D958	17.0	40	28.0	Do.
No. 7.....	D890	12.0	29		Do.
	D895	12.0	29		Do.
	D967	12.0	29	29.0	Do.
No. 8.....	D891	18.0	43		Do.
	D892	12.0	29		Do.
	D940	15.0	36		Do.
	D941	14.0	33	35	Do.

a Full length of file, 42 inches.

The evidence of no detonation in tests D887, D888, and D889 was that nothing but the noise of the detonator was audible when the trials were made.

In tests M231 and M234 an 8-inch cartridge was used.

In no trial was more than 18 inches of the 42 inches detonated. The part that detonated, in general, varied directly with the grade of the detonator.

The number of partial detonations with each detonator was as follows:

Number of partial detonations in rate-of-detonation tests with a 35 per cent strength gelatin dynamite (2 years old).

Grade of electric detonator.	Number of tests.	Number of tests in which partial detonation occurred.	Percentage of partial detona- tions.
			<i>Per cent.</i>
No. 3.....	1	1	100
No. 4.....	2	2	100
No. 5.....	4	2	50
No. 6.....	3	2	67
No. 7.....	3	3	100
No. 8.....	4	4	100

Except with the No. 3 and the No. 4 electric detonators, the number of tests with which was small, the percentage of partial detonations increased with the grade of the electric detonator.

TESTS WITH A 40 PER CENT STRENGTH GELATIN DYNAMITE, FROZEN.

The results of tests with a 40 per cent strength gelatin dynamite (frozen) follow. The diameter of the cartridges used was $1\frac{1}{2}$ inches.

Results of rate-of-detonation tests with a 40 per cent strength gelatin dynamite (frozen).

Grade of electric detonator.	Test No.	Rate of detonation.			
		First quarter.	Second quarter.	Second half.	Total.
		<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>
No. 3.....	D 1087	3 inches blown off.			
No. 4.....	D 1088	4, 018	6, 420	5, 890	5, 376
No. 5.....	D 1089	6, 250	7, 258	5, 769	6, 207
No. 6.....	D 1093	4, 167	7, 750	6, 522	5, 921
No. 7.....	D 1094	4, 687	6, 420	5, 769	5, 591
No. 8.....	D 1095	4, 018	7, 500	6, 522	5, 806
Grand average.....		4, 628	7, 075	6, 094	5, 780

* No detonation occurred with the No. 3 electric detonator.

The grand averages show that the maximum rate occurred in the second quarter, with a subsequent falling off in the rate; moreover, each individual test showed similar results, irrespective of the grade of the electric detonator used.

The variation of 14.4 per cent in the average rate is rather high, and is seemingly due to the fact that results with frozen explosives are always erratic.

Complete detonation occurred in each test with each of the six electric detonators except the No. 3, which failed to detonate.

TESTS WITH A 35 PER CENT STRENGTH GELATIN DYNAMITE
3 YEARS OLD.

The results of tests with a 35 per cent strength gelatin dynamite (three years old) follow. The cartridges used were $1\frac{1}{2}$ inches in diameter.

Results of rate-of-detonation tests with 35 per cent strength gelatin dynamite (3 years old).

Grade of electric detonator.	Test No.	Remarks.
No. 3.....	M228	No detonation.
No. 4.....	M238	Do.
No. 5.....	D888	Do.
No. 7.....	D886	Do.
No. 8.....	D885	Do.

The explosive was so old and insensitive to detonation that for the purpose of discriminating between grades of electric detonators it was useless, because in no test did detonation occur. No tests were made with the No. 6 electric detonator.

SMALL LEAD BLOCK TESTS.*

The lead blocks used in the small lead block tests were squirted with a diameter of $1\frac{1}{2}$ inches and were cut to a length of $2\frac{1}{2}$ inches. An annealed steel disk $1\frac{1}{2}$ inches in diameter and one-quarter inch high was placed above each block and above this was placed the 100-gram charge of the explosive, held in position by a paper sleeve wrapped around the block and the disk and extending above them. The electric detonator used was centrally placed in the top of the charge. When the explosion was fired, the block rested on a firm horizontal steel base. The compression of the block was determined by measuring the difference in the height of the block before and after firing.

TESTS WITH A 20 PER CENT "STRAIGHT" NITROGLYCERIN DYNAMITE WITH 6 PER CENT OF ADDED WATER.

The results of tests of a 20 per cent "straight" nitroglycerin dynamite follow. The explosive contained 6 per cent of added water:

Results of small lead block tests with a 20 per cent "straight" nitroglycerin dynamite containing 6 per cent of added water.

Grade of electric detonator.	Test No.	Com- pression.	Average com- pression.
		Mm.	Mm.
No. 3.....	B755	14.00	14.08
	B764	14.25	
	B773	14.00	
No. 4.....	B756	15.00	14.67
	B765	14.50	
	B774	14.50	
No. 5.....	B757	14.25	14.08
	B766	14.00	
	B775	14.00	
No. 6.....	B761	15.00	15.00
	B770	15.00	
	B779	15.00	
No. 7.....	B762	15.25	15.25
	B771	14.75	
	B780	15.75	
No. 8.....	B763	15.50	15.50
	B772	15.75	
	B781	15.25	

The No. 8 electric detonator produced a compression 9.6 per cent greater than that of the No. 3 electric detonator; in general with the explosive tested, the compression increased with the grade of the detonator. The No. 4 electric detonator, however, developed more energy than did the No. 5.

* For a more extended description of the small lead block test, see Bull. 15, Bureau of Mines: Investigations of explosives used in coal mines; with a chapter on the natural gas used at Pittsburgh, by G. A. Burrell, and an introduction by C. E. Monroe, by Clarence Hall, W. O. Snelling, and S. P. Howell, 1912, pp. 113-114.

TESTS WITH A 20 PER CENT "STRAIGHT" NITROGLYCERIN DYNAMITE,
FROZEN AND CONTAINING LESS THAN 6 PER CENT OF ADDED WATER.

The results of tests with a 20 per cent "straight" nitroglycerin dynamite (frozen and containing no added water or 2.5 or 4 per cent of added water) are tabulated below:

Results of small lead block tests of a 20 per cent "straight" nitroglycerin dynamite (frozen and containing less than 6 per cent of added water).

Grade of electric detonator.	Test No.	Temperature of frozen explosive.	Percentage of added water.	Compression.	Average compression.
		°C.		Mm.	Mm.
No. 3.....	B616	+2.0	0	14.25	13.33
	B622	-1.0	2.5	13.25	
	B647	-9.0	4.0	12.50	
No. 4.....	B617	+2.0	0	14.50	13.43
	B623	-1.0	2.5	13.25	
	B648	-9.0	4.0	12.50	
No. 5.....	B618	+2.0	0	13.50	13.00
	B624	-1.0	2.5	13.00	
	B649	-9.0	4.0	12.50	
No. 6.....	B619	+2.0	0	13.50	13.25
	B625	-1.0	2.5	13.25	
	B653	-9.0	4.0	13.00	
No. 7.....	B620	+2.0	0	15.00	13.53
	B625	-1.0	2.5	13.50	
	B654	-9.0	4.0	13.00	
No. 8.....	B621	+2.0	0	15.25	13.92
	B627	-1.0	2.5	13.25	
	B655	-9.0	4.0	13.25	

The tests showed the tendency of the electric detonators to increase slightly in explosive efficiency with the grade, but again the No. 3 and the No. 4 electric detonators showed an increase over the No. 5 and even over the No. 6.

TESTS WITH A 20 PER CENT "STRAIGHT" NITROGLYCERIN DYNAMITE,
FROZEN AND CONTAINING 6 PER CENT OF ADDED WATER.

As no failures had occurred with any of the electric detonators, when tested with the 20 per cent "straight" nitroglycerin dynamite, a sample of that explosive with 6 per cent of added water was frozen (temperature 9° C.) and was tested, with results as follows:

Results of small lead block tests with a 20 per cent "straight" nitroglycerin dynamite (frozen and containing 6 per cent of added water).

Grade of electric detonator.	Test No.	Compression.	Average compression.
		Mm.	Mm.
No. 3.....	B728	∞ 0.00	0.00
	B737	∞.00	
	B746	∞.00	
No. 4.....	B729	∞.00	.00
	B738	∞.00	
	B747	∞.00	

∞ Incomplete detonation.

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Results of small lead block tests with a 20 per cent "straight" nitroglycerin dynamite (frozen and containing 6 per cent of added water)—Continued.

Grade of electric detonator.	Test No.	Compression.	Average compression.
No. 5.....	B730 B739 B748	Mm. a 0.00 a.00 a.00	0.00
No. 6.....	B734 B743 B752	a.00 a.00 a.00	
No. 7.....	B735 B744 B753	12.75 13.75 a.50	
No. 8.....	B736 B745 B754	12.75 13.75 a 1.00	9.17

a Incomplete detonation.

The number of complete detonations with each detonator was as follows:

Number of complete detonations with a 20 per cent "straight" nitroglycerin dynamite (frozen and containing 6 per cent of added water).

Grade of electric detonator.	Number of tests.	Number of tests in which complete detonation occurred.	Percentage of complete detonations.
No. 3.....	3	0	0
No. 4.....	3	0	0
No. 5.....	3	0	0
No. 6.....	3	0	0
No. 7.....	3	2	67
No. 8.....	3	2	67

The explosive was very insensitive and complete detonation occurred only with the No. 7 and No. 8 electric detonators and with them in only two out of three trials with each.

TESTS WITH A 40 PER CENT STRENGTH AMMONIA DYNAMITE WITH 6 PER CENT OF ADDED WATER.

The results of tests with a 40 per cent strength ammonia dynamite with 6 per cent of added water are tabulated below:

Results of small lead block tests with a 40 per cent strength ammonia dynamite containing 6 per cent of added water.

Grade of electric detonator.	Test No.	Compression.	Average compression.
No. 3.....	B656 B665 B674 B683 B692	Mm. 7.25 8.50 9.50 8.25 7.75	8.25

Results of small lead block tests with a 40 per cent strength ammonia dynamite containing 6 per cent of added water—Continued.

Grade of electric detonator.	Test No.	Compression.	Average compression.
		<i>Mm.</i>	<i>Mm.</i>
No. 4.....	B657	7.25	8.20
	B666	8.75	
	B675	8.50	
	B684	8.00	
	B693	8.50	
No. 5.....	B658	6.00	7.70
	B667	7.25	
	B676	8.75	
	B685	8.00	
	B694	8.50	
No. 6.....	B662	9.75	8.95
	B671	8.75	
	B680	9.25	
	B689	7.75	
	B698	9.25	
No. 7.....	B663	8.00	9.15
	B672	8.75	
	B681	10.00	
	B690	10.25	
	B699	8.75	
No. 8.....	B664	8.75	9.70
	B673	9.50	
	B682	10.75	
	B691	9.75	
	B700	9.75	

This explosive showed a marked tendency to be erratic both with the higher and with the lower grades of electric detonators.

The explosive efficiency of the electric detonators increased with the grade of the electric detonator, except that the efficiency of the No. 5 electric detonator was considerably low and that of the No. 3 a trifle high.

TESTS WITH A 40 PER CENT STRENGTH GELATIN DYNAMITE, FROZEN.

Following are the results (Pl. V, A) of tests with a 40 per cent strength gelatin dynamite that was in a frozen condition:

Results of small lead block tests with a 40 per cent strength gelatin dynamite (frozen).

Grade of electric detonator.	Test No.	Temperature of frozen explosive.	Compression.	Average compression.
		<i>° C.</i>	<i>Mm.</i>	<i>Mm.</i>
No. 3.....	B633	-2.5	3.00	14.25
	B638	-5.0	16.75	
	B701	+0.5	13.00	
	B710	+2.5	13.50	
	B719	+2.5	13.75	
No. 4.....	B629	-4.5	1.50	12.62
	B639	-5.0	15.50	
	B702	+0.5	10.75	
	B711	+2.5	11.00	
	B720	+2.5	13.25	
No. 5.....	B630	-4.5	1.00	13.12
	B640	-5.0	17.25	
	B703	+0.5	13.75	
	B712	+2.5	10.75	
	B721	+2.5	10.75	

• Incomplete detonation.

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Results of small lead block tests with a 40 per cent strength gelatin dynamite (frozen)—Con.

Grade of electric detonator.	Test No.	Temperature of frozen explosive.	Compression.	Average compression.
		<i>° C.</i>	<i>Mm.</i>	<i>Mm.</i>
No. 6.....	B631	-4.5	12.50	14.95
	B644	-5.0	19.25	
	B707	+0.5	14.25	
	B716	+2.5	14.25	
	B725	+2.5	14.50	
No. 7.....	B632	-4.5	14.50	17.00
	B645	-5.0	20.25	
	B708	+0.5	18.00	
	B717	+2.5	16.25	
	B726	+2.5	16.00	
No. 8.....	B637	-2.5	15.75	17.80
	B646	-5.0	18.00	
	B709	+0.5	19.75	
	B718	+2.5	17.25	
	B727	+2.5	18.25	

The results were very erratic. The strength of the detonators increased with the grade of the electric detonator used, as shown by the average compression, except that the compression with the No. 3 electric detonator was comparatively high.

The number of complete detonations with each detonator was as follows:

Number of complete detonations in small lead block tests with a 40 per cent strength gelatin dynamite (frozen).

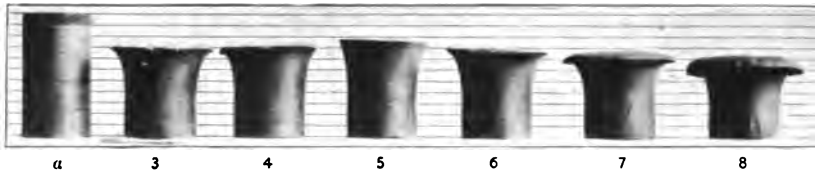
Grade of electric detonator.	Number of tests.	Number of tests in which complete detonation occurred.	Percentage of complete detonations.
			<i>Per cent.</i>
No. 3.....	5	4	80
No. 4.....	5	4	80
No. 5.....	5	4	80
No. 6.....	5	5	100
No. 7.....	5	5	100
No. 8.....	5	5	100

The results tabulated above indicate that the tendency to complete detonation increases with the grade of the electric detonator used.

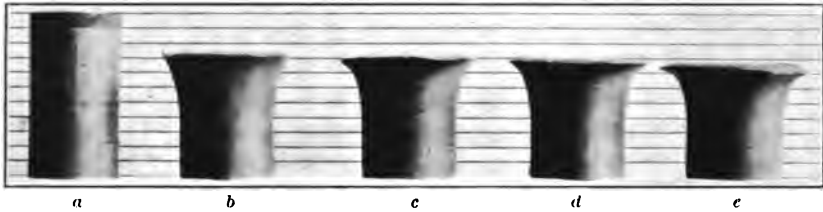
EXPLOSION-BY-INFLUENCE TESTS.^a

Explosion-by-influence tests were conducted by placing two cartridges of an explosive at a definite distance apart; each cartridge was in a vertical position, one being directly above the other. The electric detonator was placed in the lower end of the lower cartridge, so that the lower cartridge on detonation either did or did not cause

^a For a more extended description of the test, see Bull. 15, Bureau of Mines: Investigations of explosives used in coal mines; with a chapter on the natural gas used at Pittsburgh, by G. A. Burrell, and an Introduction by C. E. Munroe, by Clarence Hall, W. O. Snelling, and S. P. Howell, 1912, p. 100.



A. RESULTS OF SMALL LEAD BLOCK TESTS OF P. T. S. S. ELECTRIC DETONATORS NOS. 3, 4, 5, 6, 7, AND 8. *a*, BLOCK BEFORE TEST.



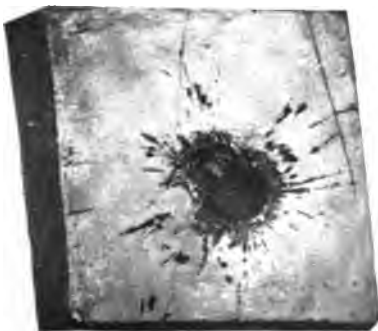
B. RESULTS OF SMALL LEAD BLOCK TESTS OF NO. 6 ELECTRIC DETONATORS. *a*, BLOCK BEFORE TEST; *b*, WESTERN COAST; *c*, SPECIAL; *d*, P. T. S. S.; *e*, FOREIGN.



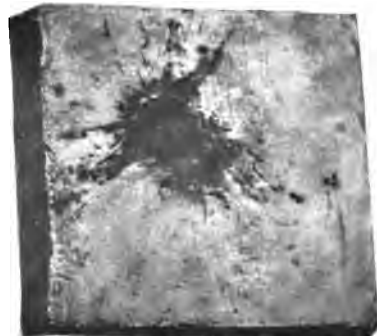
WESTERN COAST.



SPECIAL.



P. T. S. S.



FOREIGN.

C. SCORING OF LEAD PLATES BY FOUR NO. 6 ELECTRIC DETONATORS LAID ON SIDE.

detonation of the upper cartridge. The separating distance, established by successive trials, was but 1 inch greater than that at which the upper cartridge would detonate. With one explosive, however, certain trials were run with the cartridges separated by a given distance, and the number of times that the upper cartridge did or did not detonate was recorded.

TESTS WITH AN EXPLOSIVE OF CLASS 1, SUBCLASS *a*.

The results of tests with an explosive of class 1, subclass *a* (an ammonium-nitrate explosive containing a sensitizer that is itself an explosive), are tabulated below. The average weight of the cartridges was 166 grams and they measured $1\frac{1}{4}$ by 8 inches.

Results of explosion-by-influence tests with an explosive of class 1, subclass a.

Grade of electric detonator.	Test No.	Distance separating cartridges.	Result on upper cartridge.	Distance established at—
		<i>Inches.</i>		<i>Inches.</i>
No. 3.....	J874	3	Did not explode...	4
	J875	2	Exploded.....	
	J876	3	do.....	
	J877	4	Did not explode...	
	J878	4	do.....	
No. 4.....	J870	2	Exploded.....	4
	J871	3	do.....	
	J872	4	Did not explode...	
	J873	4	do.....	
No. 5.....	J764	2	do.....	2
	J765	1	Exploded.....	
	J766	2	Did not explode...	
No. 6.....	J741	4	do.....	3
	J742	3	do.....	
	J743	2	do.....	
	J744	1	Exploded.....	
	J745	2	do.....	
	J746	3	Did not explode...	
No. 7.....	J747	3	do.....	3
	J748	2	Exploded.....	
	J749	3	Did not explode...	
No. 8.....	J750	3	do.....	3
	J751	2	Exploded.....	
	J752	3	Did not explode...	

These tests did not discriminate as to the relative efficiency of the different grades of electric detonators; the efficiency of the low-grade electric detonators was at least as great as that of the high-grade electric detonators.

TESTS WITH AN EXPLOSIVE OF CLASS 4.

Following are the results of tests with an explosive of class 4 (an explosive in which the characteristic material is nitroglycerin). Except for the trials under test J896, the average weight of each cartridge was 161 grams and the size of each was $1\frac{1}{4}$ by 8 inches. In the trials under test J896 the lower cartridge weighed 161 grams and the upper one weighed 110 grams, being only 5 inches long. In all

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of the tests in which the distance separating cartridges was 5 inches, the bottoms of the cartridges (as packed) faced each other, whereas in all of the tests in which the separating distance was 4 inches, the tops of the cartridges faced each other.

Results of explosion-by-influence tests with an explosive of class 4.

Grade of electric detonator.	Test No.	Distance separating cartridges.	Result—upper cartridge.
		<i>Inches.</i>	
No. 3.....	J895	5	Did not explode.
	J895	5	Do.
	J895	5	Exploded.
	J895	5	Did not explode.
	J895	4	Do.
	J895	4	Do.
No. 4.....	J895	5	Do.
	J895	5	Do.
	J895	5	Do.
	J895	5	Exploded.
	J895	4	Do.
	J895	4	Do.
No. 5.....	J895	5	Do.
	J895	5	Do.
	J895	5	Do.
	J895	5	Exploded.
	J895	4	Did not explode.
	J895	4	Do.
No. 6.....	J895	5	Do.
	J895	5	Do.
	J895	5	Do.
	J895	5	Exploded.
	J895	4	Do.
	J895	4	Do.
No. 7.....	J895	5	Do.
	J895	5	Do.
	J895	5	Do.
	J895	5	Exploded.
	J895	4	Do.
	J895	4	Do.
No. 8.....	J895	5	Do.
	J895	5	Do.
	J895	5	Do.
	J895	5	Do.
	J895	4	Do.
	J895	4	Exploded.
	J895	4	Do.

The following tabulation shows the number of explosions of the upper cartridge:

Percentage of explosions of the upper cartridge in explosion-by-influence tests with an explosive of class 4.

Grade of electric detonator.	Number of tests.	Number of explosions of the second cartridge.	Percentage of explosions.
No. 3.....	7	1	14
No. 4.....	7	3	43
No. 5.....	7	1	14
No. 6.....	7	3	43
No. 7.....	7	3	43
No. 8.....	7	2	29

TESTS WITH A 40 PER CENT STRENGTH AMMONIA DYNAMITE CONTAINING NITROSUBSTITUTION COMPOUNDS.

Following are tabulated the results of tests with a 40 per cent strength ammonia dynamite containing nitrosubstitution compounds. The cartridges used were $1\frac{1}{4}$ by 8 inches, their average weight being 226 grams.

Results of explosion-by-influence tests with a 40 per cent strength ammonia dynamite containing nitrosubstitution compounds.

Grade of electric detonator.	Test No.	Distance separating cartridges.	Result on upper cartridge.	Distance established at—
		<i>Inches.</i>		<i>Inches.</i>
No. 5.....	J708	8	Exploded.....	9
	J709	9	Did not explode...	
	J710	9	do.....	
No. 6.....	J689	14	do.....	8
	J690	12	do.....	
	J691	9	do.....	
	J692	7	do.....	
	J693	6	do.....	
	J694	4	Exploded.....	
	J695	5	do.....	
	J696	6	do.....	
	J697	7	do.....	
No. 7.....	J698	8	Did not explode...	9
	J699	8	do.....	
	J720	9	do.....	
No. 8.....	J721	8	Exploded.....	10
	J722	9	Did not explode...	
	J704	8	Exploded.....	
	J705	9	do.....	
	J706	10	Did not explode...	
	J707	10	do.....	

No tests made with detonators Nos. 3 and 4.

TESTS WITH A 35 PER CENT STRENGTH GELATIN DYNAMITE 2 YEARS OLD.

Following are the results of tests with a 35 per cent strength gelatin dynamite (two years old). The average weight of each cartridge was 265 grams and the size of each $1\frac{1}{4}$ by 8 inches.

Results of explosion-by-influence tests with a 35 per cent strength gelatin dynamite (2 years old).

Grade of electric detonator.	Test No.	Distance separating cartridges.	Result on upper cartridge.
		<i>Inches.</i>	
No. 6.....	J724	6	Did not explode.
	J725	5	Do.
	J726	4	Do.
	J727	2	Do.
	J728	0	Do.
	J729	0	Do.
No. 7.....	J730	0	Do.
	J731	0	Do.
No. 8.....	J732	0	Do.
	J733	0	Do.

No tests were made with the No. 3, the No. 4, or the No. 5 electric detonators.

The tests failed to discriminate between the different grades of electric detonators, except to the limited extent that in two trials the lower cartridge failed to detonate completely once with the No. 6. In no trial did the detonation of the lower cartridge cause the detonation of the upper cartridge.

PERCENTAGES OF DETONATIONS IN INDIRECT TESTS OF P. T. S. S. ELECTRIC DETONATORS.

The percentages of detonations in the indirect tests of the P. T. S. S. electric detonators are given below. The percentages of detonations in the tests of each electric detonator are also averaged, each average percentage having a value proportional to the number of tests from which computed; that is, each percentage is multiplied by the number of tests it represents, and the sum of the products is divided by the total number of tests of the electric detonator considered.

Percentages of detonations in indirect tests of P. T. S. S. electric detonators.

Class of explosive.	Kind of test.	Grade of electric detonator.											
		No. 3.		No. 4.		No. 5.		No. 6.		No. 7.		No. 8.	
		Percentage of detonations.	Number of tests.	Percentage of detonations.	Number of tests.	Percentage of detonations.	Number of tests.	Percentage of detonations.	Number of tests.	Percentage of detonations.	Number of tests.	Percentage of detonations.	Number of tests.
Class 1, subclass b.	Rate of detonation.	Per cent. 0	5	Per cent. 0	5	Per cent. 0	5	Per cent. 40	5	Per cent. 100	5	Per cent. 100	5
Class 1, subclass a.	do.....	67	3	100	2	50	2	100	2	100	2	100	2
40 per cent strength gelatin dynamite (frozen).do.....	do.....	0	1	100	1	100	1	100	1	100	1	100	1
35 per cent strength gelatin dynamite (two years old).do.....	do.....	100	1	100	2	50	4	67	3	100	3	100	4
20 per cent "straight" nitroglycerin dynamite (containing 6 per cent of added water and frozen).do.....	Small lead block.	0	3	0	3	0	3	0	3	67	3	67	3
40 per cent strength gelatin dynamite (frozen).do.....	do.....	80	5	80	5	80	5	100	5	100	5	100	5
Total number of tests			18		18		20		19		19		20
Average percentage of detonations.		38.9		50.0		40.0		63.2		94.7		95.0	

COMPARATIVE EXPLOSIVE EFFICIENCY.

The percentages of explosive efficiency of the different types of P. T. S. S. electric detonators were obtained by averaging all tests in which the rate of detonation or compression was determined for all the electric detonators. The percentages of the individual electric detonators were also averaged, each average percentage having a value proportional to the number of tests from which computed; that is, each percentage is multiplied by the number of tests it represents, and the sum of the products is divided by the total number of tests of the electric detonator considered. In each case the percentage of explosive efficiency of the No. 6 electric detonator is assigned a value of 100 and is used as the unit of comparison.

Average explosive efficiency of electric detonators Nos. 3, 4, and 5 in 55 tests, 90.4.
A average explosive efficiency of electric detonators Nos. 6, 7, and 8 in 64 tests, 101.6.

COMPARATIVE EXPLOSIVE EFFICIENCY OF P. T. S. S. ELECTRIC DETONATORS.

The tabulation below shows the comparative explosive efficiency (fig. 3) of the six grades of P. T. S. S. electric detonators:

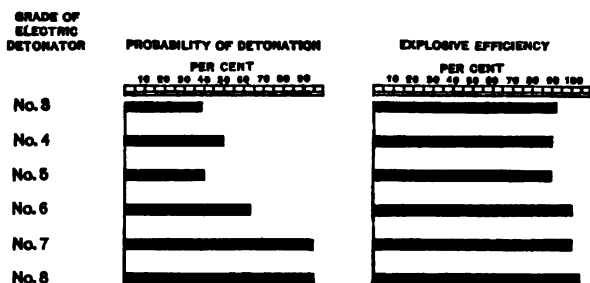


FIGURE 3.—Comparative explosive efficiency of six grades of P. T. S. S. electric detonators as determined by indirect tests.

Comparative explosive efficiency of six grades of P. T. S. S. electric detonators.

Grade of electric detonator.	Probabil- ity of detonation.	Explosive efficiency for those tests in which detonation occurred.
	Per cent.	Per cent.
No. 3.....	38.9	92.1
No. 4.....	50.0	89.6
No. 5.....	40.0	89.4
No. 6.....	63.2	100.0
No. 7.....	94.7	100.0
No. 8.....	95.0	104.5

TESTS OF FOUR NO. 6 ELECTRIC DETONATORS OF DIFFERENT MAKES.

In the tests of P. T. S. S. electric detonators as described above the composition of the fulminating charge was practically the same in each electric detonator, although there was variation in the weight of the charge. In the tests, the results of which are tabulated below, four No. 6 electric detonators manufactured by different makers were used. The weight of charge of each of the No. 6 electric detonators tested was approximately 1 gram, but each electric detonator had a different composition. The electric detonators were representative of all electric detonators used in the United States, and the tests made are of especial importance for the reason that they established for each electric detonator the charge equivalent to the Pittsburgh testing station standard electric detonators.

PHYSICAL EXAMINATION.

A physical examination was made of each of the four electric detonators (fig. 4), the results being given in the following tabulation. Each item represents an average of measurement of five electric detonators.

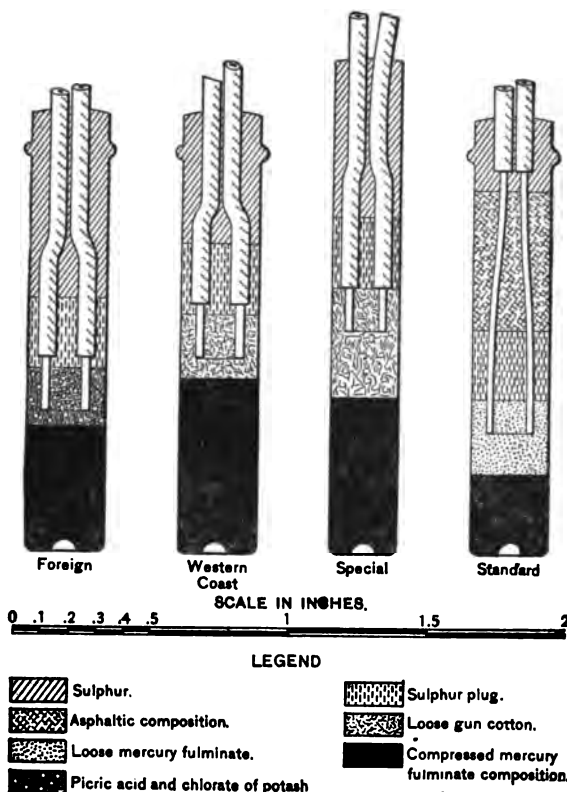


FIGURE 4.—Cross-sectional view of four No. 6 electric detonators of different makes.

Results of physical examination of four No. 6 electric detonators of different makes.

Kind of electric detonator.	Length of shell.	Outside diameter of shell.	Inside diameter of shell.	Thickness of shell.	Length of compressed charge.	Length of priming charge.	Length of sulphur plug.	Length of asphaltic composition, if any.	Length of sulphur filling.	Distance wires project below sulphur plug.	Distance from end of insulation to end of wires.
	In.	In.	In.	In.	In.	In.	In.	In.	In.	In.	In.
Western Coast.....	1.55	0.274	0.260	0.007	0.62	0.23	0.25		0.45	0.16	0.19
Special.....	1.75	.234	.220	.007	.56	.39	.25		.30	.16	.16
P. T. S. S.....	1.55	.274	.260	.007	.28	.27	.25	0.50	.25	.12	.94
Foreign.....	1.55	.274	.260	.007	.44	.21	.25		.65	.16	.19

WEIGHT AND COMPOSITION OF CHARGES.

Following is a tabulation presenting the weight of the charges and their chemical composition as determined by analysis:

RESULTS OF CALORIMETER TESTS.

The results of calorimeter tests of the four kinds of No. 6 electric detonator are tabulated below.

Results of calorimeter tests of four No. 6 electric detonators.

Kind of electric detonator.	Number of electric detonators used in each test.	Number of tests averaged.	Average heat evolved per electric detonator.	Total charge per electric detonator.	Average heat evolved per electric detonator had each contained the same weight of a charge consisting of 77.7 per cent of mercury fulminate and 22.3 per cent of chlorate of potash (exact combustion). ^a
			<i>Large calories.</i>	<i>Grams.</i>	<i>Large calories.</i>
Western Coast.....	15	2	b 0.95	0.8682	0.61
Special.....	15	2	.75	.9283	.66
P. T. S. S.....	15	2	.62	.9995	.71
Foreign.....	15	3	c 1.12	1.1748	.83

^a Berthelot, M., Explosives and their power, 1892, p. 470.

^b This unusually high value is partly due to the high heat of total combustion of nitrocellulose (about three times that of mercury fulminate).

^c This unusually high value is partly due to the high heat of total combustion of picric acid (about four times that of mercury fulminate).

SQUIRTED LEAD BLOCK TESTS.

The results of the squirted lead block tests are given herewith.

Results of squirted lead block tests^a of four No. 6 electric detonators.

Kind of electric detonator.	Test No.	Volume of bore hole.		Increase of volume.	Average increase of volume.	Weight of total charge.
		Before test.	After test.			
		<i>C. c.</i>	<i>C. c.</i>	<i>C. c.</i>	<i>C. c.</i>	<i>Grams.</i>
Western Coast.....	{ AA14 AA15	1.7 1.7	27.7 28.7	26.0 27.0	26.5	0.8682
Special.....	{ AA 9 AA41	1.4 1.5	20.6 19.3	19.2 18.8	19.0	.9283
P. T. S. S.....	{ AA10 AA11	1.7 1.7	20.0 19.8	18.3 18.1	18.2	.9995
Foreign.....	{ AA12 AA26	1.7 1.7	28.9 28.6	27.2 26.9	27.0	1.1748

^a For a description of the procedure in these tests, see p. 20.

CAST LEAD BLOCK TESTS.

Following are tabulated the results (Pl. VI) of cast lead block tests of the four kinds of No. 6 electric detonators:

Results of cast lead block tests of four No. 6 electric detonators.

Kind of electric detonator.	Test No.	Volume of bore hole.		Increase of volume.	Average increase of volume.	Weight of total charge.
		Before test.	After test.			
		<i>C. c.</i>	<i>C. c.</i>	<i>C. c.</i>	<i>C. c.</i>	<i>Grams.</i>
Western Coast.....	{ AA 6 AA31	1.8 1.7	22.7 23.2	20.9 21.5	21.2	0.8682
Special.....	{ AA33 AA38	1.4 1.4	16.1 15.2	14.7 13.8	14.2	.9283
P. T. S. S.....	{ AA30 AA35	1.7 1.6	16.0 15.2	14.3 13.6	14.0	.9905
Foreign.....	{ AA 5 AA54	1.7 1.7	19.7 20.0	18.0 18.3	18.2	1.1748

TESTS WITH LEAD PLATES.

Two series of tests of the four No. 6 electric detonators were made by the use of $\frac{1}{4}$ -inch lead plates. In one series the electric detonators were placed on end on the plates and were detonated, the resultant depression of the plates being recorded. In the other series each electric detonator was placed on its side on the lead plate before detonation.

DETONATORS ON END.

The results of the lead-plate tests in which the detonators were placed on end (Pl. VII) are tabulated below:

Results of lead-plate tests of four No. 6 electric detonators, detonators being placed on end.

Kind of electric detonator.	Test No.	Volume of water contained in depression.	Diameter of crater.	Depth of crater.	Height of cone on bottom.
		<i>C. c.</i>	<i>Mm.</i>	<i>Mm.</i>	<i>Mm.</i>
Western Coast.....	M155	0.15	11	5	Slight.
Special.....	M159	.25	11	6	2
P. T. S. S.....	M149	.40	13	7	3
Foreign.....	M153	.45	13	7	3

Results of lead-plate tests of four grades of electric detonators, detonators being placed on side.

Kind of electric detonator.	Test No.	Volume of water contained in depression.	Diameter of crater.	Depth of crater.	Height of cone on bottom.
Western Coast.....	M156	0.45	26	12	4
Special.....	M160	.50	19	11	4
P. T. S. S.....	* M150	.50	21	13	5
Foreign.....	M154	.50	22	13	5

* Bottom of plate slightly raised; not raised in other tests.



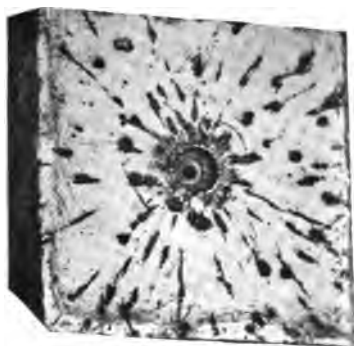
RESULTS OF CAST LEAD BLOCK TESTS OF FOUR NO. 6 ELECTRIC DETONATORS. *a*, LEAD BLOCK BEFORE TEST; *b*, WESTERN COAST; *c*, SPECIAL; *d*, P. T. S. S.; *e*, FOREIGN.



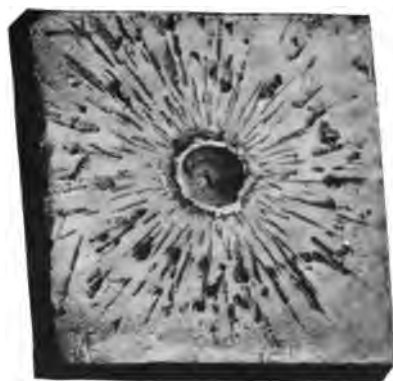
WESTERN COAST.



SPECIAL.



P. T. S. S.



FOREIGN

SCORING OF LEAD PLATES BY FOUR NO. 6 ELECTRIC DETONATORS PLACED ON END.

DETONATORS ON SIDE.

Following are tabulated the results when the detonators were placed on their side (Pl. V, *C*) on the lead plates before detonation:

A second series of tests with the $\frac{1}{2}$ -inch lead plates, the electric detonators being fired on their side, was made, and the resultant depressions of the plates were measured with sand. The results are tabulated below:

Depression of $\frac{1}{2}$ -inch lead plates when electric detonators were fired on their side, depression measured with sand.

Kind of electric detonator.	Test No.	Plate No.	Weight of sand contained in depression, measurement No.—					Average.	Grand average.	Volume, ^a
			1	2	3	4	5			
Western Coast....	M307	1	<i>Grams.</i> 0.535	<i>Grams.</i> 0.565	<i>Grams.</i> 0.544	<i>Grams.</i> 0.551	<i>Grams.</i> 0.553	<i>Grams.</i> 0.550	0.540	0.380
		2	.591	.601	.602	.590	.602	.591		
		3	.476	.470	.499	.472	.486	.478		
Special.....	M307	1	.507	.557	.562	.540	.587	.551	.556	.392
		2	.551	.563	.566	.573	.590	.569		
		3	.540	.536	.568	.542	.551	.547		
P. T. S. S.....	M301	1	.574	.565	.620	.620	.590	.594	.596	.420
		2	.580	.586	.607	.599	.596	.588		
		3	.622	.600	.615	.601	.598	.607		
Foreign.....	M307	1	.635	.668	.678	.684	.573	.648	.595	.419
		2	.602	.584	.594	.587	.610	.595		
		3	.580	.580	.540	.511	.520	.542		

^a The volume was computed from the grand average by dividing this by the specific gravity of the sand—in this case 1.42.

The results of the tests are fairly satisfactory, as they practically agree with the explosive efficiency established for electric detonators by the indirect methods.

NAIL TESTS.

The nail tests previously described were also used in connection with the investigation of the four grades of No. 6 electric detonators. The results (Pl. IV, *B*) are tabulated below:

Results of nail tests of four No. 6 electric detonators.

Kind of electric detonator.	Test No.	Angle of bending resulting from trial No.—					Average.	Minimum.
		1	2	3	4	5		
Western Coast.....	M286	•	•	•	•	•	•	•
Special.....	M287	22	24	20	28	27	24.2	20
P. T. S. S.....	M288	16	35	17	16	23	21.4	16
Foreign.....	M300	23	24	25	24	26	24.4	23
		17	31	18	19	18	20.6	17

RATE-OF-DETONATION TESTS.

Rate-of-detonation tests similar to those with the different grades of P. T. S. S. electric detonators were conducted with the four No. 6 electric detonators. The results, according to the explosive used, are presented below.

TESTS WITH AN EXPLOSIVE OF CLASS 1, SUBCLASS *a*.

Following are the results of tests with an explosive of class 1, subclass *a* (an ammonium-nitrate explosive containing a sensitizer that is itself a sensitizer). The diameter of the cartridges used was seven-eighths of an inch.

Results of rate-of-detonation tests with an explosive of class 1, subclass a.

Kind of electric detonator.	Test No.	Rate of detonation in tube.							
		First quarter.		Second quarter.		Second half.		Full length.	
		Individual rate.	Average rate.	Individual rate.	Average rate.	Individual rate.	Average rate.	Individual rate.	Average rate.
		<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>
Western Coast.....	{ D970 D971	{ 2,343 2,296	2,319	{ 2,296 2,260	2,272	{ 2,585 2,556	2,570	{ 2,445 2,406	2,426
Special.....	{ D976 D978 D979	{ 1,891 2,419 2,393	2,234	{ 2,296 2,393 2,206	2,296	{ 2,761 2,469 2,419	2,546	{ 2,368 2,432 2,356	2,385
P. T. S. S.	{ D966 D967	{ 2,445 2,472	2,458	{ 2,678 2,596	2,632	{ 2,205 2,205	2,205	{ 2,368 2,356	2,362
Foreign.....	{ D974 D973	{ 2,778 2,143	2,460	{ 1,814 2,396	2,104	{ 2,795 2,966	2,830	{ 2,450 2,528	2,494
Grand average.....			2,393		2,326		2,538		2,417

The average rate for the meter length was practically uniform, but such difference as was shown indicated that the ascending order of explosive efficiency of the detonators is as follows: P. T. S. S., special, Western Coast, foreign.

The percentage of complete detonations with each detonator was as follows:

Percentage of complete detonations with an explosive of class 1, subclass a.

Kind of electric detonator.	Number of tests.	Number of tests in which incomplete detonation occurred.	Percentage of complete detonations.
Western Coast.....	2	0	100
Special.....	5	0	100
P. T. S. S.	2	0	100
Foreign.....	3	1	67

TESTS WITH AN EXPLOSIVE OF CLASS 1, SUBCLASS *b*.

Two rate-of-detonation tests were made of each of the four kinds of No. 6 electric detonators on an explosive of class 1, subclass *b* (an ammonium-nitrate explosive containing a sensitizer that is not itself an explosive) being used. The cartridges used were $1\frac{1}{4}$ inches in diameter. In no test did detonation occur, so that the tests failed to discriminate between the different kinds of electric detonators.

TESTS WITH A 20 PER CENT "STRAIGHT" NITROGLYCERIN DYNAMITE.

The results of tests with a 20 per cent "straight" nitroglycerin dynamite are presented below. The diameter of the cartridges was seven-eighths of an inch.

Results of rate-of-detonation tests with a 20 per cent "straight" nitroglycerin dynamite.

Kind of electric detonator.	Test No.	Rate of detonation in tube.							
		First quarter.		Second quarter.		Second half.		Full length.	
		Individual rate.	Average rate.	Individual rate.	Average rate.	Individual rate.	Average rate.	Individual rate.	Average rate.
		<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>
Western Coast.....	{ D906 D907	2,778 3,462	3,120	{ 2,960 2,679	2,820	{ 3,147 3,285	3,216	{ 3,000 3,147	3,074
Special.....	{ D908 D909	3,048 3,399	3,174	{ 2,928 2,587	2,758	{ 3,089 3,027	3,048	{ 3,027 2,967	2,997
P. T. S. S.....	{ D1000 D1002	3,729 3,947	3,838	{ 2,683 2,443	2,563	{ 2,767 3,309	3,038	{ 2,933 3,158	3,046
Foreign.....	{ D994 D996	2,922 3,000	2,961	{ 3,125 2,557	2,841	{ 2,866 3,061	2,964	{ 2,941 2,903	2,922
Grand average.....			3,273		2,746		3,066		3,010

The figures representing the grand averages indicate that the rate was influenced by a negative acceleration in the second quarter meter followed by a positive acceleration in the second half meter. This acceleration occurred in all tests except D996 and D994.

The uniformity of the rates for the last half meter and the meter is noteworthy.

With a 20 per cent "straight" nitroglycerin dynamite such difference as was shown in the tests indicated that the ascending order of explosive efficiency is: Foreign, special, P. T. S. S., Western Coast.

TESTS WITH A 40 PER CENT STRENGTH AMMONIA DYNAMITE CONTAINING NITROSUBSTITUTION COMPOUNDS.

Following are tabulated the results of tests with a 40 per cent ammonia dynamite containing nitrosubstitution compounds. The explosive was repacked in cartridges seven-eighths of an inch in diameter.

54 INVESTIGATIONS OF DETONATORS AND ELECTRIC DETONATORS.

Results of rate-of-detonation tests with a 40 per cent strength ammonia dynamite containing nitrosubstitution compounds.

Kind of electric detonator.	Test No.	Rate of detonation in tube.							
		First quarter.		Second quarter.		Second half.		Full length.	
		Individual rate.	Average rate.	Individual rate.	Average rate.	Individual rate.	Average rate.	Individual rate.	Average rate.
		<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>
Western coast.....	{ D913 D923 }	2,557 3,082 }	2,820 }	{ 2,394 2,500 }	2,447 }	{ 2,812 2,616 }	2,714 }	{ 2,632 2,687 }	2,660 }
Special.....	{ D879 D914 }	2,830 2,587 }	2,704 }	{ 2,588 2,781 }	2,684 }	{ 2,900 3,048 }	2,974 }	{ 2,794 2,853 }	2,824 }
P. T. S. S.....	{ D904 D919 D920 }	2,368 2,472 2,250 }	2,363 }	{ 2,298 3,041 2,446 }	2,594 }	{ 2,572 2,663 2,543 }	2,593 }	{ 2,446 2,696 2,439 }	2,527 }
Foreign.....	{ D912 D922 }	2,250 2,616 }	2,433 }	{ 3,129 3,125 }	3,127 }	{ 2,446 2,557 }	2,502 }	{ 2,828 2,542 }	2,535 }

The rate for the second quarter meter was the highest for the P. T. S. S. and the foreign electric detonators.

With a 40 per cent strength ammonia dynamite containing nitrosubstitution compounds the tests indicated that the ascending order of explosive efficiency is: P. T. S. S., foreign, Western Coast, special.

TESTS WITH A 35 PER CENT STRENGTH GELATIN DYNAMITE 2 YEARS OLD.

The results of tests with a 35 per cent strength gelatin dynamite (two years old) are tabulated below. The diameter of the cartridges used was 1½ inches.

Results of rate-of-detonation tests with a 35 per cent strength gelatin dynamite two years old.

Kind of electric detonator.	Test No.	Length of part of file blown off or detonated.	Percentage of file that detonated.	Average.	Remarks.
		<i>Inches.</i>	<i>Per cent.</i>	<i>Per cent.</i>	
Western Coast.....	{ D897 D898 }	7.0 8.0 }	17 19 }	18.0 }	{ Partial detonation. Do.
Special.....	{ D901 D902 }	10.0 9.0 }	24 21 }	22.5 }	{ Do. Do.
P. T. S. S.....	{ D889 D898 D958 }	7.0 18.0 17.0 }	0 43 40 }	27.5 }	{ No detonation. Partial detonation. Do.
Foreign.....	{ D899 D900 }	6.0 7.0 }	14 17 }	15.5 }	{ Do. Do.

^a Full length of file, 42 inches.

The evidence of no detonation in test D889 was that nothing but the noise of the electric detonator was audible when the trial was made.

With the two-year-old sample of 35 per cent strength gelatin dynamite used the tests indicated that the ascending order of explosive efficiency is: Foreign, Western Coast, special, P. T. S. S.

The percentage of partial detonations with each electric detonator was as follows:

Percentage of partial detonations with a 35 per cent strength gelatin dynamite two years old.

Kind of electric detonator.	Number of tests.	Number of tests in which partial detonation occurred.	Percentage of complete detonations.
			<i>Per cent.</i>
Western Coast.....	2	2	100
Special.....	2	2	100
P. T. S. S.....	3	2	67
Foreign.....	2	2	100

TESTS WITH A 40 PER CENT STRENGTH GELATIN DYNAMITE, FROZEN.

Following are the results of tests with a 40 per cent strength gelatin dynamite (frozen). The diameter of the cartridges used was $1\frac{1}{4}$ inches.

Results of rate-of-detonation tests with a 40 per cent strength gelatin dynamite, frozen.

Kind of electric detonator.	Test No.	Rate of detonation in tube.			
		First quarter.	Second quarter.	Second half.	Full length.
		<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>	<i>Meters per second.</i>
Western Coast.....	D1091	3, 273	7, 177	5, 705	5, 028
Special.....	D1092	4, 167	6, 357	6, 013	5, 460
P. T. S. S.....	D1093	4, 167	7, 760	6, 522	5, 921
Foreign.....	D1090	3, 090	14, 933	5, 361	5, 235
Grand average.....		3, 674	9, 032	5, 900	5, 411

The figures included in the "grand average" show that the maximum rate occurred in the second quarter, with a subsequent falling off in the rate; moreover, the rate varied similarly in each individual test.

With the explosive used in the tests the results indicate that the ascending order of explosive efficiency is: Western Coast, foreign, special, P. T. S. S.

TESTS WITH A 35 PER CENT STRENGTH GELATIN DYNAMITE 3 YEARS OLD.

One test each of the Western Coast, the special, and the foreign No. 6 electric detonators was made with a 35 per cent strength gelatine dynamite 3 years old. The diameter of the cartridges used was $1\frac{1}{4}$ inches. No detonation took place in any of the tests, as the explosive was so old and insensitive to detonation that for the purpose of discriminating between detonators it was useless.

SMALL LEAD BLOCK TESTS.

Small lead block tests were made with the four No. 6 electric detonators. The results, according to the explosive tested, are given below.

TESTS WITH A 20 PER CENT "STRAIGHT" NITROGLYCERIN DYNAMITE.

Three series of tests were conducted with a 20 per cent "straight" nitroglycerin dynamite in different conditions as indicated below.

Results of small lead block tests with a 20 per cent "straight" nitroglycerin dynamite with 6 per cent of added water.

Kind of electric detonator.	Test No.	Compression.	Average compression.
		<i>Mm.</i>	<i>Mm.</i>
Western Coast.....	{ B759 B768 B777	{ 15.00 14.75 15.25	{ 15.00
Special.....	{ B760 B769 B778	{ 14.00 15.25 15.25	{ 14.83
P. T. S. S.....	{ B761 B770 B779	{ 15.00 15.00 15.09	{ 15.00
Foreign.....	{ B758 B767 B776	{ 14.25 14.75 15.00	{ 14.67

In the tests the explosive produced nearly uniform individual compressions and little difference in the average compressions, but such difference as was shown indicated that the ascending order of explosive efficiency is: Foreign, special, Western Coast, P. T. S. S.

The results of tests with the same explosive, but containing 4 per cent of added water and frozen, were as follows:

Results of small lead-block tests with a 20 per cent "straight" nitroglycerin dynamite containing 4 per cent of added water and frozen.

Kind of electric detonator.	Test No.	Temperature of frozen explosive.	Percentage of water added.	Compression.	Average compression.
		<i>°C.</i>		<i>Mm.</i>	<i>Mm.</i>
Western Coast.....	B651	-9.0	4.0	12.75	12.75
Special.....	B652	-9.0	4.0	12.50	12.50
P. T. S. S.....	B653	-9.0	4.0	13.00	13.00
Foreign.....	B650	-9.0	4.0	12.75	12.75

With the explosive in the condition mentioned, the results of the tests indicate that the ascending order of explosive efficiency is: Special, Western Coast, foreign, and P. T. S. S.

Further tests were conducted with 6 per cent of water added to the explosive and the explosive frozen (temperature 9° C.). Three tests were made with each of the four grades of electric detonators, but the explosive was too insensitive to detonation to be discriminative, as no compression of any of the blocks was produced.

TESTS WITH A 40 PER CENT STRENGTH AMMONIA DYNAMITE.

Following are the results of tests with a 40 per cent ammonia dynamite, to which had been added 6 per cent of water:

Results of small lead block tests with a 40 per cent strength ammonia dynamite containing 6 per cent of added water.

Kind of electric detonator.	Test No.	Compression.	Average compression.
		Mm.	Mm.
Western Coast	B660	8.25	8.75
	B669	9.00	
	B678	9.00	
	B687	8.00	
	B686	9.50	
Special.....	B661	8.00	8.40
	B670	8.25	
	B679	8.50	
	B688	7.75	
	B697	9.25	
P. T. S. S.....	B662	9.75	8.95
	B671	8.75	
	B680	9.25	
	B689	7.75	
	B696	9.25	
Foreign.....	B659	8.00	9.20
	B668	10.00	
	B677	9.25	
	B686	9.50	
	B695	9.25	

The results were obviously erratic. However, the tests indicated that the ascending order of explosive efficiency is: Special, Western Coast, P. T. S. S., foreign.

TESTS WITH A 40 PER CENT STRENGTH GELATIN DYNAMITE, FROZEN.

The results of tests with a 40 per cent strength gelatin dynamite (frozen) were as follows (Pl. V, B):

Results of small lead block tests with a 40 per cent strength gelatin dynamite, frozen.

Kind of electric detonator.	Test No.	Temperature of frozen explosive.	Compression.	Average compression.
		° C.	Mm.	Mm.
Western Coast	B635	-2.5	14.75	13.90
	B642	-5.0	17.75	
	B705	+ .5	12.50	
	B714	+2.5	12.50	
	B723	+2.5	12.00	
Special.....	B636	-2.5	12.75	14.85
	B643	-5.0	13.00	
	B706	+ .5	15.00	
	B715	+2.5	14.75	
	B724	+2.5	13.75	
P. T. S. S.....	B631	-4.5	12.50	14.95
	B644	-5.0	19.25	
	B707	+ .5	14.25	
	B716	+2.5	14.25	
	B725	+2.5	14.50	
Foreign.....	B634	-2.5	11.00	15.15
	B641	-5.0	13.00	
	B704	+ .5	15.00	
	B713	+2.5	16.25	
	B722	+2.5	15.50	

As indicated by the table, the results of the tests were very erratic with this frozen gelatin dynamite. The insensitiveness of this explosive has been mentioned in a foregoing section regarding the incompleteness of detonation in tests with the Nos. 3, 4, and 5 electric detonators. With the No. 6 electric detonators, however, detonation was complete in every trial.

EXPLOSION-BY-INFLUENCE TESTS.

Tests involving explosion by influence as outlined in a foregoing section relative to tests of different grades of P. T. S. S. electric detonators were made of the four kinds of No. 6 electric detonators, as described below:

TESTS WITH AN EXPLOSIVE OF CLASS 1, SUBCLASS *a*.

Following are the results of tests with an explosive of class 1, subclass *a* (an ammonium-nitrate explosive containing a sensitizer that is itself an explosive). The size of the cartridges used was 1½ by 8 inches and the average weight was 166 grams.

Results of explosion-by-influence tests with an explosive of class 1, subclass a (sample 1).

Kind of electric detonator.	Test No.	Distance separating cartridges.	Result on upper cartridge.	Established distance at which detonation did not occur.
		<i>Inches.</i>		<i>Inches.</i>
Western Coast.....	J758	2	Did not explode.....	2
	J759	1	Exploded.....	
	J760	2	Did not explode.....	
Special.....	J761	1	Exploded.....	2
	J762	2	Did not explode.....	
	J763	2	do.....	
P. T. S. S.....	J741	4	do.....	3
	J742	3	do.....	
	J743	2	do.....	
	J744	1	Exploded.....	
	J745	2	do.....	
	J746	3	Did not explode.....	
Foreign.....	J753	3	do.....	1
	J754	2	do.....	
	J755	1	do.....	
	J756	0	Exploded.....	
	J757	1	Did not explode.....	

With the ammonium-nitrate explosive used, the results of the tests indicated that the ascending order of explosive efficiency is: Foreign, Western Coast and special, P. T. S. S.

TESTS WITH AN EXPLOSIVE OF CLASS 4.

The results of tests with an explosive of class 4 (an explosive in which the characteristic material is nitroglycerin) are tabulated below. The size of the cartridges used was 1½ by 8 inches, their average weight being 161 grams, except that in the trials under test J896 the upper cartridge weighed 110 grams and was 5 inches long.

Results of explosion-by-influence tests with an explosive of class 4.

Kind of electric detonator.	Test No.	Distance separating cartridges.	Result on upper cartridge.
		<i>Inches.</i>	
Western Coast.....	J895	5	Exploded.
	J895	5	Did not explode.
	J895	5	Do.
	J895	5	Do.
	J896	4	Exploded.
	J896	4	Do.
	J896	4	Did not explode.
Special.....	J895	5	Exploded.
	J895	5	Did not explode.
	J895	5	Do.
	J895	5	Do.
	J896	4	Exploded.
	J896	4	Do.
	J896	4	Did not explode.
P. T. S. S.....	J895	5	Do.
	J895	5	Do.
	J895	5	Do.
	J895	5	Exploded.
	J896	4	Do.
	J896	4	Do.
	J896	4	Did not explode.
Foreign.....	J895	5	Do.
	J895	5	Do.
	J895	5	Do.
	J895	5	Do.
	J896	4	Do.
	J896	4	Do.
	J896	4	Do.

Percentage of explosions of the upper cartridge in explosion-by-influence tests with an explosive of class 4.

Kind of electric detonator.	Number of tests.	Number of explosions of upper cartridge.	Percentage of explosions of upper cartridge.
Western Coast.....	7	3	<i>Per cent.</i> 43
Special.....	7	3	43
P. T. S. S.....	7	3	43
Foreign.....	7	0	0

In all tests in which the distance separating cartridges was 5 inches the bottoms of the cartridges (as packed) faced each other; in all tests in which the distance was 4 inches the tops of the cartridges faced each other.

The tests indicated that the foreign electric detonator was not as effective under the conditions of the tests as were the other three.

TESTS WITH A 40 PER CENT STRENGTH AMMONIA DYNAMITE CONTAINING NITROSUBSTITUTION COMPOUNDS.

Following are the results of tests with a 40 per cent strength ammonia dynamite containing nitrosubstitution compounds. The cartridges used measured $1\frac{1}{4}$ by 8 inches and their average weight was 226 grams.

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Results of explosion-by-influence tests with a 40 per cent strength ammonia dynamite containing nitrosubstitution compounds.

Kind of electric detonator.	Test No.	Distance separating cartridges.	Result on upper cartridge.	Distance at which detonation did not occur.
		<i>Inches.</i>		<i>Inches.</i>
Western Coast.....	J714	8	Exploded.....	9
	J715	9	Did not explode.....	
	J716	9	do.....	
Special.....	J717	8	do.....	8
	J718	7	Exploded.....	
	J719	8	Did not explode.....	
P. T. S. S.....	J689	14	do.....	8
	J690	12	do.....	
	J691	9	do.....	
	J692	7	do.....	
	J693	6	do.....	
	J694	4	Exploded.....	
	J695	5	do.....	
	J696	6	do.....	
	J697	7	do.....	
	J698	8	Did not explode.....	
	J699	8	do.....	
Foreign.....	J711	8	do.....	8
	J712	7	Exploded.....	
	J713	8	Did not explode.....	

The tests show practically the same result regardless of the electric detonator used.

TESTS WITH A 35 PER CENT STRENGTH GELATIN DYNAMITE 2 YEARS OLD.

The results of tests with a 35 per cent strength gelatin dynamite (two years old) are tabulated below. The cartridges used were 1½ by 8 inches, their average weight being 265 grams.

Results of explosion-by-influence tests with a 35 per cent strength gelatin dynamite (two years old).

Kind of electric detonator.	Test No.	Distance separating cartridges.	Result on upper cartridge.
		<i>Inches.</i>	
Western Coast.....	J784	0	Did not explode.
	J785	0	Do.
Special.....	J787	0	Do.
	J788	0	Do.
P. T. S. S.....	J724	6	Do.
	J725	5	Do.
	J726	4	Do.
	J727	2	Do.
	J728	0	Do.
	J729	0	Do.
Foreign.....	J786	0	Do.
	J789	0	Do.

These tests failed to discriminate between the different detonators, as in no trial did the explosion of the lower cartridge cause the detonation of the upper cartridge.

TRAUZI LEAD BLOCK TESTS.*

In testing the four kinds of No. 6 electric detonators the Trauzl lead block tests were used in addition to the tests previously described.

The Trauzl lead blocks are cylindrical in shape, measuring 200 mm. in diameter and 200 mm. in height. They have an axial bore hole 25 mm. in diameter and 125 mm. in depth. The charge of 20 grams of the explosive in which the electric detonator was embedded was placed in the bottom of the bore hole and no stemming was used. The increase in the volume of water that the bore hole would contain after an explosion was the result recorded.

Following is a tabulation of results of Trauzl lead block tests in which a 20 per cent "straight" nitroglycerin dynamite was used. The charge of explosive in each test was 20 grams, to which was added 6 per cent of water.

Results of Trauzl lead block tests with a 20 per cent "straight" nitroglycerin dynamite.

Kind of electric detonator.	Test No.	Expansion.	Average expansion.
		<i>C. c.</i>	<i>C. c.</i>
Western Coast.....	{ A817 A818 }	{ 175 173 }	174
Special.....	{ A819 A821 }	{ 177 176 }	176
P. T. S. S.....	{ A822 A824 }	{ 178 173 }	176
Foreign.....	{ A815 A816 }	{ 178 179 }	178

As indicated by the table, the average expansion of the blocks in each test was nearly the same.

PERCENTAGES OF DETONATIONS IN INDIRECT TESTS OF FOUR KINDS OF NO. 6 ELECTRIC DETONATORS.

The percentages of detonations in the indirect tests of the four kinds of No. 6 electric detonators are given below. The percentages of detonations in the tests of each electric detonator are also averaged, each average percentage having a value proportional to the number of tests from which it is computed; that is, each percentage is multiplied by the number of tests it represents and the sum of the products is divided by the total number of tests of the electric detonator considered.

* For a more extended description of this test see Bull. 15, Bureau of Mines: Investigations of explosives used in coal mines; with a chapter on the natural gas used at Pittsburgh, by G. A. Burrell, and an introduction by C. E. Munroe, by Clarence Hall, W. O. Snelling, and S. P. Howell, 1912, pp. 114-116.

Percentages of detonations in indirect tests of four kinds of No. 6 electric detonators.

Class and grade of explosive.	Character of test.	Western Coast.		Special.		P. T. S. S.		Foreign.	
		Per-centage of deto-nations.	Num-ber of tests.	Per-centage of deto-nations.	Num-ber of tests.	Per-centage of deto-nations.	Num-ber of tests.	Per-centage of deto-nations.	Num-ber of tests.
Class 1, subclass a (sample 1).	Rate of det-onation.	Per ct. 100	2	Per ct. 100	5	Per ct. 100	2	Per ct. 67	3
40 per cent strength gelatin dynamite (frozen).	do.....	100	1	100	1	100	1	100	1
35 per cent strength gelatin dynamite (2 years old).	do.....	100	2	100	2	67	3	100	2
20 per cent "straight" nitroglycerin dynamite (frozen and containing 6 per cent of added water).	Small lead block.	0	3	0	3	0	3	0	3
40 per cent strength gelatin dynamite (frozen).	do.....	100	5	100	5	100	5	100	5
Average.....		55.5		71.4		63.2		61.1	
Total number of tests..			18		21		19		18

COMPARATIVE EXPLOSIVE EFFICIENCY.

The percentage of explosive efficiency of the four kinds of No. 6 electric detonators was obtained by averaging all tests in which the rate of detonation, compression, or expansion was determined for all detonators. Each percentage was given a value proportional to the number of tests from which the percentage was computed. In each case the percentage of explosive efficiency of the P. T. S. S. No. 6 electric detonator is given a value of 100 and is taken as the unit of comparison.

Explosive efficiency of four No. 6 electric detonators of different makes.

Class of explosive.	Kind of test.	Kind of electric detonator.																					
		Western Coast.				Special.				P. T. S. S.				Foreign.									
		Rate.	Compression.	Expansion.	Percentage of explosive efficiency.	Number of tests.	Rate.	Compression.	Expansion.	Percentage of explosive efficiency.	Number of tests.	Rate.	Compression.	Expansion.	Percentage of explosive efficiency.	Number of tests.							
20 per cent "straight" nitroglycerin dynamite in 1-inch cartridges. 40 per cent strength ammonia dynamite (containing nitrosubstitution compounds) in 1-inch cartridges. Class 1, sub class a (sample 1) in 1-inch cartridges. 40 per cent strength gelatin dynamite (frozen in 1-inch cartridges). 35 per cent strength gelatin dynamite (2 years old). 20 per cent "straight" nitroglycerin dynamite (containing 6 per cent of added water). 20 per cent "straight" nitroglycerin dynamite (containing 4 per cent of added water). 40 per cent strength gelatin dynamite (frozen). 40 per cent strength ammonia dynamite (containing 5 per cent of added water). 20 per cent "straight" nitroglycerin dynamite (containing 6 per cent of added water).	Rate-of-detonation.	Meters per sec. 3,974	Mm.	Cu. mm.	Per cent. 100.9	2	Meters per sec. 2,997	Mm.	Cu. mm.	Per cent. 98.4	2	Meters per sec. 3,046	Mm.	Cu. mm.	Per cent. 100.0	2	Meters per sec. 2,922	Mm.	Cu. mm.	Per cent. 98.9	2		
	do.	2,660			105.3	2	2,824			111.8	2	2,587			100.0	3	2,535			100.3	2		
	do.	2,428			102.7	2	2,385			101.0	3	2,362			100.0	2	2,494			105.5	2		
	do.	5,028			84.9	1	5,460			92.2	1	5,981			100.0	1	5,285			88.4	1		
	do.	a 7.5			42.9	2	a 9.5			54.3	2	a 17.5			100.0	3	a 6.5			37.1	2		
	Small lead block.	15.00			100.0	3	14.83			94.9	3	15.00			100.0	3	14.67			97.8	3		
	do.	12.75			98.1	1	12.50			96.2	1	13.00			100.0	1	12.75			98.1	1		
	do.	13.90			93.0	5	14.85			99.3	5	14.95			100.0	5	15.15			101.3	5		
	do.	8.75			97.8	5	8.40			93.9	5	8.66			100.0	5	9.20			102.8	5		
	Transit lead block.	174	98.9	2						176	100.0	2				178	101.1	2				178	101.1
Average.				93.5	26				98.5	26						95.2				95.2			
Total number of tests.				93.5	26				98.5	26						95.2				95.2			

a Inches that detonated.

COMPARATIVE EXPLOSIVE EFFICIENCY OF FOUR KINDS OF NO. 6 ELECTRIC DETONATORS.

The comparative explosive efficiency of the four grades of electric detonators (fig. 5), as established, is tabulated below.

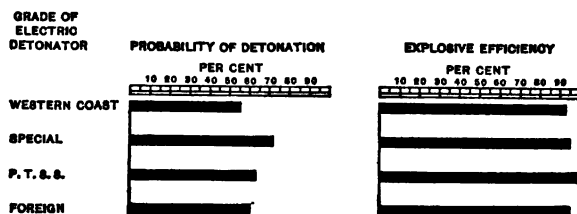


FIGURE 5.—Comparative explosive efficiency of four kinds of No. 6 electric detonators as established by indirect tests.

Explosive efficiency of four kinds of No. 6 electric detonators.

Kind of electric detonator.	Percentage of probability of detonation.	Explosive efficiency for those tests in which detonation occurred.
	<i>Per cent.</i>	<i>Per cent.</i>
Western Coast	55.5	91.5
Special	71.4	95.5
P. T. & S. No. 6	63.2	100.0
Foreign	61.1	95.2

RELATIVE STRENGTH OF DETONATORS AND ELECTRIC DETONATORS.

It is generally recognized that the safest way of firing shots in blasting operations is with electric detonators by means of the electric current. This is especially true in gaseous coal mines, because if fuse is used the flame produced by the burning fuse may ignite such inflammable gaseous mixtures as are present. There is also danger of a hangfire when the charge may be exploded unexpectedly, due to the smoldering of the fuse.

There are, however, many conditions of mining under which electric detonators can not be used advantageously. In driving drifts in many of the metal mines of this country fuse is generally used. In work of this kind it is often necessary to fire dependent shots, and the flying rock from one shot may disconnect or cause short-circuiting of the electric wires of the detonators wired for succeeding shots. When fuse is used, different lengths can be cut and, before lighting, the projecting ends can be coiled in a place within the mouth of the hole where they are well protected.

It has been observed that mercury fulminate ignited in small quantities develops its full force only when confined. It also has been believed that the sulphur plug in an electric detonator

offers more confinement to the fulminating composition of a detonator than a piece of fuse does, even when the fuse is properly used and securely crimped in place. Therefore it seemed desirable to make comparative efficiency tests of both electric detonators and detonators fitted with fuse. The nail test was adopted for the reason that it produced more nearly the results established for the efficiency of detonators than any of the other direct methods.

The nail test was made with the No. 3 and the No. 6 detonators and with the electric detonators made from these detonators, with the following results:

Results of nail tests with No. 3 and No. 6 detonators and electric detonators.

Grade of detonator or electric detonator.	Test No.	Degrees of bending in trial—					Average.	Minimum.
		1	2	3	4	5		
No. 3 detonator ^a	M289	8	7	9	9	8	8.2	7
No. 3 electric detonator.....	M279	12	10	8	9	7	9.2	7
No. 6 detonator ^a	M318	32	29	35	30	31	31.4	29
No. 6 electric detonator.....	M321	31	31	33	36	27	31.6	27

^a Fired with fuse placed against the compressed charge of mercury fulminate composition and crimped in place.

The compressed charge of the No. 6 electric detonator weighed 1.0225 grams and consisted of 89.61 per cent of mercury fulminate and 10.39 per cent of potassium chlorate. The priming charge consisted of 0.02 gram of loose guncotton. As the weight of charge of this electric detonator was practically the same as of the P. T. S. S. electric detonator, the increased strength as shown by the nail test (nail bent 31.6° by the No. 6 electric detonator as compared with 24.4° by the P. T. S. S. electric detonator) would indicate that the quantity of compressed charge in the detonator may be a function of the efficiency of the detonator.

The tests showed that with low-grade detonators the strength of electric detonators is slightly greater than that of the corresponding detonators, but that with greater charges, such as the No. 6 detonators contain, the strength of the two types is practically the same. This indicates that the additional confinement given by the plug of the electric detonator as compared with the fuse of the fuse detonator is important only with the low-grade detonators.

A serious objection to the use of fuse and detonators in wet blasting is the fact that it is impossible to perfect a waterproof seal at the top of the detonator when it is crimped on the fuse. The ordinary fuse crimper depends on flattening the sides of the copper shell to contract the diameter of the detonator. Tests have shown that when a detonator is crimped on a fuse in this manner and submerged under

water for 30 minutes the fulminating charge and the powder train at the end of the fuse in the detonator become damp. The spit of the lighted fuse, if the fuse burns through, is usually of insufficient intensity to cause an explosion of the fulminating charge. In some cases a sharp explosion or an explosion of a very low order occurs. If only a little water enters the detonator, the spit of the burning fuse is often sufficient to cause the fulminating charge to detonate with a sharp report and completely destroy the copper shell. In some of the tests 70 to 80 per cent of the compressed fulminating charge was recovered in the lower part of the copper shell. The spit of the burning fuse had seemingly caused a part of the fulminating composition to detonate, the detonation destroying the top part of the copper shell but not being propagated throughout the remainder of the wet fulminating charge. In these instances a slight report only was audible. Obviously an explosion of this order would not cause a complete detonation of dynamite or other high explosives. In some of the tests a thin coating of tallow was placed on the fuse one-fourth of an inch from the end and extending a distance of one-half of an inch up on the fuse. In the tests in which tallow was placed around the fuse before it was inserted into the detonator a more perfect seal was made.

A new crimper recently placed on the market crimps the detonator on the fuse in a manner different from that of any of the other types of crimpers. The salient feature of this crimper is its ability to contract the top of the detonator uniformly and to form a $\frac{1}{8}$ -inch groove around the copper shell, thus perfecting a seal of the detonator on the fuse that will permit submersion under water for 30 minutes. The shell is pressed firmly and uniformly into the fuse, but not far enough to break or separate the powder train. Owing to the varying diameters of different types of fuse and the probability of considerable variation in the same type or even in the same coil of fuse, the use of a thin film of tallow around the end of the fuse that is inserted into the detonator, as described above, will make a better seal irrespective of the crimper used.

TESTS WITH A TRINITROTOLUENE DETONATING FUSE.

As the results of all tests made with explosives sensitive to detonation showed that when a complete detonation was obtained the rate of detonation was practically the same, the authors decided to carry on tests with a few explosives, using No. 6 electric detonators and trinitrotoluene detonating fuse as the initiatory explosive.

The trinitrotoluene detonating fuse used in the tests was a lead tube filled with trinitrotoluene, and is commercially known as "cordeau detonant." The results of physical examination of the fuse were as follows.

Results of physical examination of 6-mm. detonating fuse (cordeau detonant).

Outside diameter, inches.....	0.2275
Thickness of lead, inches.....	.0275
Inside diameter of tube, inches.....	.1725
Weight of a foot length, grams.....	41.74
Weight of a foot length of lead tube, grams.....	35.32
Weight of a foot length of charge, grams.....	6.42
Density of charge.....	1.40
Consistency of charge: Powdered; very fine; dry; soft; slightly cohesive.	
Color of charge: Straw.	

The tests were made with explosives in which a 6-inch length of the detonating fuse (cordeau detonant) was embedded centrally at one end of the charge, the side of the fuse being slit and spread open from one end a distance of $1\frac{1}{2}$ inches. A No. 6 electric detonator was placed against the trinitrotoluene in the slit and tied firmly in place. The electric detonator and attached detonating fuse (cordeau detonant) was imbedded in the explosive. Following is a tabulation of the results of the tests:

Results of rate-of-detonation tests of explosives with a No. 6 electric detonator and detonating fuse (cordeau detonant).

Class of explosive.	Test No.	Rate of detonation.	Average rate of detonation.
		<i>Meters per second.</i>	<i>Meters per second.</i>
Class 1, subclass a (sample 1).....	D861 D862	2,231 2,225	2,228
Class 1, subclass b (sample 2).....	D890 D891	(a) (a)	
20 per cent "straight" nitroglycerin dynamite.....	D1007 D1008 D1009	2,156 2,947 3,190	3,098
40 per cent strength ammonia dynamite (containing nitrosubstitution compounds).....	D880 D883 D884 D885 D886	2,444 (a) 2,045 2,821 2,713	
35 per cent strength gelatin dynamite (two years old).....	D893 D894	(a) (a)	

a Incomplete detonation; rate not determined.

Comparative results of tests with detonating fuse fired with No. 6 detonators and with No. 6 electric detonators used alone.

	No. 6 electric detonator and detonating fuse.	No. 6 electric detonator.
Averages of the rate of detonation of three explosives, meters per second.....	2,686	2,645
Explosive efficiency, per cent.....	101.6	100.0

The results of the tests show that a 6-inch length of detonating fuse (cordeau detonant) used in connection with a No. 6 electric detonator does not increase the rate of detonation of the explosives

tested. The slight increase indicated in the table is explained by the fact that the rate of detonation of detonating fuse (cordeau detonant) itself is about 4,900 meters per second and that the fuse extended about one-eighth the length of the charge.

If the detonating fuse had extended the full length of the charge the rate of detonation of the explosive would probably have been increased to 4,900 meters per second, the rate of the detonating fuse.

Detonating fuse has been used to some extent in deep-hole blasting. A piece of the fuse is laid beside the charge of high explosive that has been inserted into the hole. The fuse, when detonated, accelerates the rate of detonation of the explosive, thus producing a greater shattering effect on the surrounding rock. Detonating fuse has also been used to replace electric detonators in large blasts when simultaneous blasting is desired. Obviously, when detonating fuse is used in drill holes containing a long charge of explosive whose rate of detonation is less than that of the detonating fuse, a greater shattering effect will be produced. When the rate of detonation of the explosive charge is greater than that of the detonating fuse, the only advantage in using the fuse would be to insure a complete detonation of the entire charge of explosive.

TESTS WITH DETONATORS DISTRIBUTED IN CHARGE.

With long charges of high explosives in blasting work, it has sometimes been the custom to place detonators at intervals in the charges, in the belief that the work accomplished by the explosives would be increased. Rate-of-detonation tests were made with an explosive of class 1, subclass *a*, sample 3 (an ammonium-nitrate explosive containing a sensitizer that is itself an explosive), in charges $1\frac{1}{2}$ inches in diameter, with and without No. 7 detonators distributed in the explosive, to determine whether detonation of the charge would occur at a greater distance because of the presence of the detonators. The explosive had been previously tested (see p. 29) in charges $1\frac{1}{2}$ inches in diameter, with the result that the No. 7 electric detonator caused complete detonation in every trial, whereas the No. 3 electric detonator failed to do so once out of three trials. The explosive was insensitive to detonation and was purposely chosen for this reason. The results were as follows:

Results of rate-of-detonation tests in which No. 7 detonators were distributed in the charge.

Grade of detonator.	Test No.	Dimensions of galvanized-iron tube used.		Result.
		Diameter.	Length.	
No. 7.....	^a D1134	<i>Inches.</i> 1½	42	Detonation incomplete. Do. Do.
No. 7.....	^b D1147	1½	80	
No. 7.....	^c D1148	1½	42	

^a No detonators distributed in the charge.

^b Three No. 7 detonators placed every one-half meter in the charge.

^c Three No. 7 detonators placed every one-fourth meter in the charge.

Further tests were made with an insensitive gelatin dynamite by placing one No. 7 detonator every one-eighth meter in the charge, with results as follows:

Results of rate-of-detonation tests in which No. 7 detonators were distributed in the charge.

Grade of detonator.	Diameter of cartridges.	Dimensions of galvanized-iron tube used.		Length of charge that detonated.
		Diameter.	Length.	
	<i>Inches.</i>	<i>Inches.</i>	<i>Inches.</i>	<i>Inches.</i>
No. 7.....	1½	1½	42	21½
No. 7.....	1½	1½	42	16
No. 7.....	1½	1½	42	20

The results of rate-of-detonation tests with the same explosive, when no extra detonators were used, were as follows:

Results of rate-of-detonation tests without extra detonators.

Grade of detonator.	Diameter of cartridge.	Dimensions of galvanized-iron tube used.		Length of charge that detonated.
		Diameter.	Length.	
	<i>Inches.</i>	<i>Inches.</i>	<i>Inches.</i>	<i>Inches.</i>
No. 7.....	1½	1½	42	6
No. 7.....	1½	1½	42	15
No. 7.....	1½	1½	42	9

The results of the tests tabulated above indicate that extra detonators distributed 5 inches apart in a cartridge file of an insensitive explosive 40 inches long have a slight tendency to increase the propagation of the explosive wave, but that extra detonators placed 10 inches apart offer no advantage.

With an insensitive gelatin dynamite, such as that used in the tests, the influence of the detonator probably does not extend further than 5 inches. Furthermore, the explosion wave and the detonation of the explosive surrounding the detonator probably precede the explosion of the detonator. Assuming this to be true, the detonator is exploded in the products of combustion of the explosive, and, accordingly, offers little, if any, advantage as a means of extending the explosive wave.

Attention is called, however, to the fact that it is often advantageous to fire simultaneously two or more electric detonators placed in different parts of the charge in the same drill hole. Under these conditions, if the charge is fired simultaneously, the time of detonation would be reduced, and, accordingly, the shattering effect of the explosion would be materially increased. Also when long charges of explosives are used, it is sometimes necessary to use more than one electric detonator in the charge to insure complete detonation.

In quarry operations the large drill-hole method of blasting is being rapidly introduced. The former practice of quarrying by the bench method and drilling holes of small diameter, which in many cases requires the chambering of the bottom of the drill holes before loading the main charge, is more expensive.

In some quarries 6-inch holes are drilled 100 feet in depth, and several thousand pounds of explosive is used in a blast. The charges usually extend to a distance of 30 feet up from the bottom of the holes. It has been found that when one electric detonator is placed in the top of the charge it will not always produce a complete detonation of the entire charge in the drill hole; therefore two or more electric detonators distributed throughout the charge are generally used. When the most violent effect is desired in blasting, the best method of placing electric detonators in a charge 30 feet in length, irrespective of whether they are connected in series or parallel, is to place one electric detonator 5 feet from the bottom of the charge, one 5 feet below the top of the charge, and one in the center of the charge. Assuming that the entire charge detonates at a uniform rate, if the three electric detonators are fired simultaneously it can readily be seen that the duration of the explosive reaction will be one-sixth of the time that would be required if one electric detonator were used in the top of the charge.

Tests were made at the bureau's Pittsburgh testing station to determine whether simultaneous explosion would occur when four of the P. T. S. S. No. 6 electric detonators were connected in series and fired with different sources of electric current. In the tests in which a 4-hole firing machine of the dynamo-electric type was used, the time interval that elapsed between the firing of the first and the last electric detonator of each series varied from 0.0004 to 0.0050 second. As it requires only 0.0020 second for 30 feet of 40 per cent "straight" nitroglycerin dynamite to detonate, it is obvious that in many cases the only advantage in using more than one electric detonator in the same charge, when fired with a 4-hole firing machine, would be to insure complete detonation of the charge. It is to be noted that the time interval between the firing of the first and the last electric detonator is in some cases greater than the time required for 30 feet of 40 per cent dynamite to detonate. When a 4-hole firing machine is used to fire four electric detonators connected in series there is not sufficient current generated to fuse the platinum wires in the electric detonators. The wires are brought to different temperatures, depending on their cross-sectional area and, accordingly, the ignition of the priming charge in the electric detonators is not simultaneous, nor is its burning or detonation uniform. These causes are assumed to be responsible for the delay that occurs in the explosion of the electric detonators.

Further tests were made by using a 10-hole firing machine, all other conditions being the same as in the previous tests. The machine furnished ample current to fuse the platinum wires in the electric detonators and they were therefore fired practically simultaneously. The time interval was only 0.0001 second. In order to obtain the benefit of simultaneous blasting when two or more electric detonators are to be fired, the source of electric current should be such as to insure the instantaneous fusing of all the bridges in the electric detonators. This can be best accomplished in practical operations by wiring all electric detonators in parallel and using a light or power circuit for firing. If a high-pressure alternating current is the only source of electricity available, it may be necessary to install a transformer in order to obtain the proper pressure without injury to the leading wires. A lamp bank or a short length of fuse wire is sometimes placed in the electric circuit and answers the same purpose, irrespective of the kind or the pressure of the current supplied. However, if a lamp bank or a fuse wire is used, it should have a greater current-carrying capacity than is necessary to fire all of the electric detonators.

USE OF TWO KINDS OF EXPLOSIVES IN THE SAME DRILL HOLE.

In certain quarry operations in the Middle West, owing to variations in the hardness and structure of the different strata, it is necessary to use more than one kind of explosive in the same drill hole. The part of the drill hole that penetrates the hardest stratum is usually loaded with an explosive having a high rate of detonation. The remainder of the charge may be an explosive having an intermediate rate. In some cases black blasting powder is used, provided there are no pronounced clay seams or other irregularities that would allow the gases evolved on the explosion of the black blasting powder to escape before the main charge detonated. In work of this kind, the holes are drilled vertically 15 to 20 feet deep, and there is always sufficient stemming used to insure the maximum effect of the blast, even when the explosives used in the same drill hole detonate at different rates.

The practice of using combination charges of explosives has been recently adopted in some coal mines. The drill holes are usually shallow and, accordingly, do not permit the use of sufficient stemming properly to confine the gases when they are evolved at different rates. Under such conditions fires and blown-out shots are likely to result.

Several tests made at the Pittsburgh testing station to determine the energy developed by combination charges showed that there was no advantage in using them under conditions that simulated blasting in coal. In some of the tests, a No. 6 detonator (blasting cap) was

inserted in the charge of dynamite, and placed in the back of the bore hole. In front of the detonator a charge of black blasting powder, containing a black powder igniter, was placed, and the free part of the drill hole was then well tamped with clay.

The results of the tests made in the ballistic pendulum, using combination charges of 40 per cent "straight" nitroglycerin dynamite and FFF black blasting powder, with and without a No. 6 detonator embedded in the explosive, were as follows:

The swings of the ballistic pendulum^a in those tests in which the detonator was used were 3.42, 3.41, 3.40, 3.41, 3.26, 3.32, 3.01, 3.34, and 3.28 inches; average, 3.32 inches. In those tests in which no detonator was used the swings were 3.58, 3.30, 3.32, 3.38, 3.24, 3.31, 3.22, 3.36, and 3.31 inches; average, 3.34 inches.

The tests indicated that there is no advantage in using an extra detonator in the dynamite, as the explosion of the black blasting powder is sufficient to cause complete detonation. Many accidents have occurred in coal mines where combination charges containing detonators were used. When squibs are used for firing, it is necessary to insert a needle into the charge of black blasting powder, and there is always then a possibility of the needle coming in contact with the detonator.*

The practice of using combination charges in coal mines offers no advantage, and, as there are many dangers attendant upon their use, the practice should be discouraged.

PUBLICATIONS ON MINE ACCIDENTS AND TESTS OF EXPLOSIVES.

The following Bureau of Mines publications may be obtained free by applying to the Director Bureau of Mines, Washington, D. C.:

BULLETIN 10. The Use of Permissible Explosives, by J. J. Rutledge and Clarence Hall. 1912. 34 pp., 5 pls.

BULLETIN 15. Investigations of Explosives Used in Coal Mines, by Clarence Hall, W. O. Snelling, and S. P. Howell, with a chapter on the natural gas used at Pittsburgh, by G. A. Burrell, and an introduction by C. E. Munroe. 1911. 197 pp., 7 pls.

BULLETIN 17. A Primer on Explosives for Coal Miners, by C. E. Munroe and Clarence Hall. 61 pp., 10 pls. Reprint of United States Geological Survey Bulletin 423.

BULLETIN 20. The Explosibility of Coal Dust, by G. S. Rice, with chapters by J. C. W. Frazer, Axel Larsen, Frank Haas, and Carl Scholz. 204 pp., 14 pls. Reprint of United States Geological Survey Bulletin 425.

BULLETIN 44. First National Mine-Safety Demonstration, Pittsburgh, Pa., October 30 and 31, 1911, by H. M. Wilson and A. H. Fay, with a chapter on the explosion at the experimental mine, by G. S. Rice. 1912. 75 pp., 7 pls.

BULLETIN 46. An Investigation of Explosion-Proof Mine Motors, by H. H. Clark. 1912. 44 pp., 6 pls.

BULLETIN 48. The Selection of Explosives Used in Engineering and Mining Operations, by Clarence Hall and S. P. Howell. 1913. 50 pp., 3 pls.

^a The ballistic pendulum used by the Bureau of Mines is a large mortar swung from a pivoted support. The explosive to be tested is fired from a small cannon into the mouth of the mortar, and the swing of the mortar is taken as a measure of the strength of the explosive.

BULLETIN 52. Ignition of Mine Gases by the Filaments of Incandescent Lamps, by H. H. Clark and L. C. Ilsley. 1913. 31 pp. 6 pls.

TECHNICAL PAPER 4. The Electrical Section of the Bureau of Mines, Its Purpose and Equipment, by H. H. Clark. 1911. 12 pp.

TECHNICAL PAPER 6. The Rate of Burning of Fuse as Influenced by Temperature and Pressure, by W. O. Snelling and W. C. Cope. 1912. 28 pp.

TECHNICAL PAPER 7. Investigations of Fuse and Miners' Squibs, by Clarence Hall and S. P. Howell. 1912. 19 pp.

TECHNICAL PAPER 11. The Use of Mice and Birds for Detecting Carbon Monoxide After Mine Fires and Explosions, by G. A. Burrell. 1912. 15 pp.

TECHNICAL PAPER 12. The Behavior of Nitroglycerin When Heated, by W. O. Snelling and C. G. Storm. 1912. 14 pp., 1 pl.

TECHNICAL PAPER 13. Gas Analysis as an Aid in Fighting Mine Fires, by G. A. Burrell and F. M. Seibert. 1912. 16 pp.

TECHNICAL PAPER 14. Apparatus for Gas-Analysis Laboratories at Coal Mines, by G. A. Burrell. 1913. 24 pp., 7 figs.

TECHNICAL PAPER 17. The Effect of Stemming on the Efficiency of Explosives, by W. O. Snelling and Clarence Hall. 1912. 20 pp.

TECHNICAL PAPER 18. Magazines and Thaw Houses for Explosives, by Clarence Hall and S. P. Howell. 1912. 34 pp., 1 pl.

TECHNICAL PAPER 19. The Factor of Safety in Mine Electrical Installations, by H. H. Clark. 1912. 14 pp.

TECHNICAL PAPER 21. The Prevention of Mine Explosions; Report and Recommendations, by Victor Watteyne, Carl Meissner, and Arthur Desborough. 12 pp. Reprint of United States Geological Survey Bulletin 369.

TECHNICAL PAPER 22. Electrical Symbols for Mine Maps, by H. H. Clark. 1912. 11 pp., 8 figs.

TECHNICAL PAPER 23. Ignition of Mine Gas by Miniature Electric Lamps, by H. H. Clark. 1912. 5 pp.

TECHNICAL PAPER 24. Mine Fires, a Preliminary Study, by G. S. Rice. 1912. 51 pp.

TECHNICAL PAPER 28. Ignition of Mine Gas by Standard Incandescent Lamps, by H. H. Clark. 1912. 6 pp.

TECHNICAL PAPER 40. Metal-Mine Accidents in the United States during the Calendar Year 1911, by A. H. Fay. 1913. 54 pp.

TECHNICAL PAPER 46. Quarry Accidents in the United States during the Calendar Year 1911, compiled by A. H. Fay. 1913. 32 pp.

TECHNICAL PAPER 48. Coal-Mine Accidents in the United States, 1896-1912, with Monthly Statistics for 1912, by F. W. Horton. 1913. 72 pp.

TECHNICAL PAPER 53. Proposed Regulations for the Drilling of Gas and Oil Wells, with Comment thereon, by O. P. Hood and A. G. Haggem. 1913. 28 pp., 2 figs.

MINERS' CIRCULAR 3. Coal-Dust Explosions, by G. S. Rice. 1911. 22 pp.

MINERS' CIRCULAR 4. The Use and Care of Mine-Rescue Breathing Apparatus, by J. W. Paul. 1911. 24 pp.

MINERS' CIRCULAR 5. Electrical Accidents in Mines; Their Causes and Prevention, by H. H. Clark, W. D. Roberts, L. C. Ilsley, and H. F. Randolph. 1911. 10 pp., 3 pls.

MINERS' CIRCULAR 6. Permissible Explosives Tested Prior to January 1, 1912, and Precautions to be Taken in Their Use, by Clarence Hall. 1912. 20 pp.

MINERS' CIRCULAR 9. Accidents from Falls of Roof and Coal, by G. S. Rice. 1912. 16 pp.

MINERS' CIRCULAR 10. Mine Fires and How to Fight Them, by J. W. Paul. 1912. 14 pp.

MINERS' CIRCULAR 11. Accidents from Mine Cars and Locomotives, by L. M. Jones. 1912. 16 pp.



Bulletin 60

**DEPARTMENT OF THE INTERIOR
BUREAU OF MINES
JOSEPH A. HOLMES, DIRECTOR**

**HYDRAULIC MINE FILLING
ITS USE IN THE
PENNSYLVANIA ANTHRACITE FIELDS
A PRELIMINARY REPORT**

BY

CHARLES ENZIAN



**WASHINGTON
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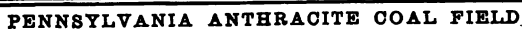
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10 0 **Scale** 10 20 Miles

After map of U. S. Geological Survey.

HYDRAULIC MINE FILLING; ITS USE IN THE PENNSYLVANIA ANTHRACITE FIELDS.

By CHARLES ENZIAN.

INTRODUCTORY STATEMENT.

This report is issued by the Bureau of Mines as one of a series dealing with methods of increasing safety and efficiency in mining operations. It is intended purely as a preliminary statement of the present development of hydraulic mine filling as conducted in the anthracite region of Pennsylvania. A discussion of different methods, their variations and modifications, as well as their probable extension as a result of amplified tests of the various roof-supporting materials, will be presented in a future bulletin.

The history of the development of the anthracite-mining industry, from the time when the Indian tribes in the Wyoming and Lackawanna Valleys first discovered the existence of "stone coal," on through subsequent events—the purchase of lands containing enormous deposits of coal for insignificant sums of money, the development of the first mine, the formation of the first company, the building of canals and railroads—up to the present time when the industry has become of such tremendous importance to the prosperity of north-eastern Pennsylvania and of the Nation itself, is intersprinkled with romance, tragedy, and pathos.

The great economic lessons to be learned from a study of mining operations in the anthracite region suggest to the person interested in the efficient utilization of natural resources the need of applying such methods as will make possible the conservation to posterity of this invaluable deposit of coal that has resulted from the operation of the processes of nature during probably hundreds of millions of years.

It is the opinion of many prominent mining men familiar with mining operations in the United States and abroad that under certain conditions the most practical way of preventing loss of unmined coal in pillars and of protecting surface property from damage by subsidence is by filling the workings with refuse material. This bulletin aims to present a general discussion of hydraulic mine filling, its origin, development, and practicability in anthracite mining.

The fundamental object of this bulletin is to enlist the thoughtful cooperation of the entire mining fraternity in such methods of mining as will accomplish the greatest good, by avoiding stream pollution, by conserving natural resources, and by reducing mining losses to a minimum. The value of mine filling as a roof support, whether for surface protection, reinforcement of pillars, or reclamation of pillars, can hardly be placed too highly. In fact, under certain geological conditions, particularly the occurrence of water-bearing strata over the coal, the maximum winning of coal practically can not be economically attained without the use of mine filling.

TERMS USED.

In this discussion the author has deviated slightly from common usage in his choice of terms, believing some of those that have been used in connection with the process of filling mine workings by hydraulic methods to be neither suitable nor descriptive. Heretofore the process has been termed and even at present is known in the Pennsylvania anthracite region as "slushing," "flushing," and "silting." As a result of various suggestions from men of long experience in this work, the name "hydraulic mine filling" (filling the mine with materials transported by water) was adopted for use in this bulletin.

The expression "physical conditions," as applied to mine workings, refers particularly to the manner of mining, whether uniform or irregular; the size of openings and the size and shape of pillars; the extent of roof and coal falls, and the general accessibility of the mine workings. The expression "geological conditions" relates particularly to the general dip of the coal bed and to local variations of dip or interruptions of continuity from folds or faults.

The term "pitch" as used in this report signifies both the inclination or dip of a coal bed, and hence the inclination of the workings in the bed, and, as in the expression "pitch workings," an inclination of more than a certain number of degrees.

A well-defined distinction exists between the terms "mine loss," "breaker loss," and "coal loss." Mine loss refers to such low-grade coal as is rejected by the miner at his working face because in his opinion it will not meet with the requirements of the rigid inspection in the breaker when his coal is dumped from the cars into the chute leading to the first picking chutes, screens, or shaker bars. Breaker loss refers to such low-grade coal as is rejected during the process of breaker preparation. Coal loss embraces both the material rejected by the miner at the face and by the inspectors at the breaker, and that which is lost, mostly by breakage, in transportation, storage, and marketing.

ACKNOWLEDGMENTS.

The author hereby acknowledges the helpful cooperation of the coal companies in the anthracite region and of their officials who have extended so many facilities for the collection of valuable data. He also acknowledges the inspiration drawn from the work of Dr. J. A. Holmes, Director of the Bureau of Mines, whose earnest efforts to promote scientific investigations in mining have inspired many others; the helpful criticism of G. S. Rice, chief mining engineer of the bureau, whose observations on European methods were of aid in shaping the text; and the valuable assistance of A. H. Fay and Samuel Sanford, engineers of the bureau, in the compilation of the data presented.

GROWTH OF THE ANTHRACITE INDUSTRY.

William Griffith* compiled a chronology of the anthracite-mining industry with great care, giving the principal events in its history from the earliest times up to and including 1908. It is particularly interesting to note that the shipping of anthracite coal began in 1776. The chronology states further:

In 1792 Col. Weiss and others formed the Lehigh Coal Mine Co., the first of its kind in the United States. In 1803 this company succeeded in getting two ark loads, about 30 tons, to market at Philadelphia, but no purchasers could be found. City authorities, as an experiment, tried to burn it beneath the boiler at the water works, but "it only served to put the fire out," and the remainder of the shipment was broken up and distributed over the sidewalks of the vicinity in place of gravel.

In 1808 Judge Jesse Fell recorded the fact that he had tried the experiment "of burning the common stone coal of the valley in a grate in a common fireplace in my house and find it will answer the purpose of fuel, making a clearer and better fire at less expense than burning wood in the common way."

The Pennsylvania anthracite region is geographically divided into four sections, generally known as the northern, eastern middle, western middle, and southern fields. The coal shipped from the various fields is known under trade names as follows: That from the northern as "Wyoming," that from the eastern middle and the eastern southern as "Lehigh," and that from the western middle and the western southern as "Schuylkill." The quantity^b of coal shipped from the various fields from 1820, when the first shipment of commercial importance (365 tons) was made from the Lehigh region, to January 1, 1912, has been as follows: Wyoming region, 959,347,466 tons; Schuylkill region, 576,745,104 tons; Lehigh region, 283,258,115 tons; aggre-

* 1908 Rept. of the Dept. of Mines of Pennsylvania, 1909, Pt. I, pp. xxx-xxxv; reprinted in the Scranton Truth, issue of Feb. 24, 1908.

^b Parker, E. W., Mineral Resources, U. S., 1911; U. S. Geol. Survey, 1911, pp. 11-12.

gating 1,819,350,685 tons. As will be noted, the respective percentages of production of the three regions are 53, 32, and 15. The estimated value of the 80,732,013 long tons of anthracite coal shipped during 1911 was \$174,852,843. The entire quantity of anthracite coal thus far shipped to market from these three regions, if loaded on one train of coal cars, would require a train long enough to encircle the globe 11 times or it would represent a prism having a base 1 mile square and a height of 2,600 feet.

HISTORICAL REVIEW OF HYDRAULIC MINE FILLING.

The first instance of the application of hydraulic mine filling is reported to have been at a mine in the Schuylkill region in 1884 for the purpose of extinguishing a mine fire at an intermediate level on the main haulage slope. After the failure of many attempts to extinguish the fire by flooding the slope with large quantities of water sent down intermittently, the superintendent conceived the idea of running the fine culm and coal dirt from the culm bank into this water, with the hope of filling up and thereby sealing the affected portion of the slope.

About 1886 and 1887 hydraulic mine filling was introduced in the Lehigh and Schuylkill regions for the purpose of sealing off mine fires, arresting squeezes, and supporting the surface. During the latter part of 1890 and the early part of 1891 hydraulic mine filling on a more systematic and economical basis was introduced in the northern field.^a This installation was practically the first in which the filling material was conveyed to the mine workings through a pipe line laid from the breaker down the shaft and into the worked-out chambers. Shortly after this time a party of foreign engineers visited Pennsylvania. The members studied hydraulic mine filling as then practiced and after returning to Europe proceeded to adopt it. The method has now become highly developed in several European coal fields and has been described repeatedly in foreign publications.

In the mines throughout the northern anthracite field of Pennsylvania hydraulic mine filling has become a part of the regular scheme of mining operations, being used particularly in areas in which inaccessible mine fires exist, or where squeezes are active, or where there is danger from surface subsidence, or where it is desired to reclaim coal pillars. Mine filling was used about 1895 to retard mine fires at the Summit Hill^b and the Sioux collieries. An early (1887) example of hydraulic mine filling for surface support is that at the

^a Davies, J. B., 1897, Rept. of the Bureau of Mines, Dept. of Internal Affairs of Pennsylvania, 1898, pp. XXXV-LI; also Pa. Anth. Operators' Assn., February, 1898.

^b Stoeck, H. H., Sealing off Summit Hill mine fire; Mines and Minerals, Aug., 1909, pp. 1-5.

Kohinoor colliery, Shenandoah, Pa. In many mines throughout the entire anthracite region the reclamation of pillars is possible to a great extent because of former hydraulic filling.

The development of hydraulic mine filling in the anthracite mines was due in a marked degree to a serious problem confronting the operators—the proper disposition of the principal by-product resulting from the mining and the preparation of anthracite coal for the market. This by-product, generally called “culm,” consists of that part of the breaker product that passes through the screens over which the smallest size of commercial product passes.

APPLICATIONS OF HYDRAULIC MINE FILLING.

The filling of mine workings may be said in general to be the outcome of a desire to get more merchantable coal from a given area of ground. Like many other useful innovations, the process originated in an attempt to meet a grave emergency. The principal applications of hydraulic filling are described below.

EXTINGUISHING MINE FIRES.

The hydraulic method of mine filling is reported to have been employed as early as 1884 to extinguish a serious mine fire. The fire originated in one of the deep lifts or levels of a haulage slope in an anthracite mine, and after raging for several days was making its way among timbers and fallen coal to higher levels, where its extinguishment by methods then in use would have been very expensive and might have permanently ruined the mine. Water had been turned down the slope in flooding quantities at regular intervals with the hope of checking and extinguishing the flames. After considerable time had been lost in this manner with no apparent improvement or success, the idea of sending down culm mixed with water was conceived and applied. The intermittent flooding with water did not fill up crevices and openings in the débris and fallen coal and rock, but after culm had been flushed into the lower section of the slope for some time it filled the interstices of the fallen material, thus excluding the air and soon bringing the fire under control. Several years later hydraulic mine filling began to be generally employed for the purpose of extinguishing or smothering isolated mine fires, either by the direct filling of the workings affected or by the construction of temporary or permanent barriers. This practice is now common and is termed “sealing off a fire by the hydraulic-filling method.”

The successful application of the various methods of extinguishing mine fires with large quantities of water, whether in steady streams, in pulsating streams, or in flooding quantities, or with mixtures of

water and refuse material, depends to a great extent on the geological structure of the coal bed and the situation of the mine workings and the mine in relation to the available supply of water and "filler." At a mine operated at a considerable distance from available filling material it may be necessary to lay long pipe lines to a convenient surface location and to furnish expensive motive power.

The filler, after having been properly prepared, must be sent into the mine through a suitable opening. At many mines only bore holes are practicable. For such mines the best location of the bore hole is determined from examination of maps or other available data; sometimes from the best recollection of old-time miners. The latter necessity arises, in the case of old workings, because the maps of such workings, made at a time when the mining engineer or surveyor was seldom considered necessary, are incomplete or unsatisfactory.

The filler, after passing down the bore hole, flows unconfined into the inaccessible workings, causing blockages among the caves, and forming finally an effective permanent sealing pillar. In some mines this requires weeks of filling, and in other mines blockage is complete in less than a day. Under more favorable conditions, as in a mine where a fire may be in progress in "live" or producing workings and where the filler can be transported to and deposited at predetermined points, the burning section is isolated so that the fire can not spread to adjacent workings, and the fire is allowed to burn to extinction within the sealed area.

The fire may be smothered by depositing filler in such a manner as to confine completely the burning district within well-defined bounds by filling the openings so as to exclude air and thus cause such a deficiency of oxygen in the atmosphere that it will not support combustion. This method was used in a Wilkes-Barre mine in which a fire had been in progress for some years. The burning area was on an anticline and the fire could not be extinguished by the usual methods of flooding. Bulkheads were constructed in the mine workings at lower points, and the open space inside of the bulkheads was filled by means of pipes which were run either above the bulkheads, along crosscuts and traveling ways, or through the bulkheads, so that they discharged some distance above the bulkheads, to insure absolutely air-tight blockage at the bulkheads.

ARRESTING MINE SQUEEZES.

The use of hydraulic filling for arresting mine squeezes was adopted many years ago shortly after its introduction for the purpose of extinguishing mine fires. In mine squeezes the attending phenomena are probably as menacing as those of mine fires. The

crunching and roaring noises of distant caves are alarming even to the most experienced miner and terrify those who live near or over the workings affected.

The first observed indications of a squeeze are a slight chipping of pillars. The usual cause of a squeeze is an excessive extraction of coal, too little being left in pillars. The quantity of coal usually extracted is determined almost entirely by local conditions, the physical character of the coal, its structure, the depth at which the bed lies, the thickness of the bed, and the method by which it is worked. An excellent discussion of the proper percentage of coal to be removed under stated conditions has been given by Bunting.*

When a squeeze has started the retarding of its progress is very difficult, especially if the bed is inclined. Pillars in inclined beds are crushed by the vertical pressure of overburden, and in addition are subjected to a side thrust through the settling of the roof. To stop a squeeze, timber is often placed in every conceivable form and position, as battalions of props and cogs, but when timber is ineffective, retreat is made to a presumably safe distance, and part of the unaffected territory ahead of the line of danger has to be sacrificed. Bulkheads are erected and filler is deposited as far as possible toward the advance of the squeeze. It is highly important that the greatest possible quantity of filler of the best quality be deposited in the shortest possible time. In emergencies like these "a minute in time may save the mine."

SUPPORTING THE SURFACE.

Hydraulic mine filling has played an important part and been highly effective in the prevention of surface subsidence. The first recorded application dates back to 1886, and the published account substantiates the assertion that hydraulic filling effectively avoided the settling of a large part of a mining town. In this instance the pillars in a moderately worked section of the mine where the dip was steep began to "run" and caused a slight subsidence of the surface. In many mines where coal has been removed somewhat excessively under valuable surface land and structures, the openings underneath are either partly filled with hydraulic filler, or "stows" of mine rock are built and the hydraulic filler is flushed in to fill the voids or interstices in the stows. In the Pennsylvania anthracite fields work of this kind is undertaken by either the surface owners or the operators, according to the provisions in the deed or lease conveying the surface. Usually the operators perform the work, the expense being borne by them or by the surface owner, as stipulated.

* Bunting, Douglas, Chamber pillars in deep anthracite mines: *Trans. Am. Inst. Min. Eng.*, vol. 42, 1911, pp. 236-245.

RECLAIMING PILLARS AND INCREASING YIELD.

A use of hydraulic filling that is practical and at the same time directly profitable was devised for the double purpose of disposing of refuse and of reclaiming pillars. The disposal of refuse, involving possible stream pollution, is discussed below; the reclamation of pillars pertains directly to the economical extraction of all the coal in favorable sections and also to the unfavorable geologic conditions due to overlying water-bearing strata. This use was introduced in the northern anthracite field in 1891.* By means of systematic filling the pillars were made available for extraction by the splitting, blocking, or slabbing methods, as the conditions required; later by refilling the rooms (chambers) the maximum yield of coal was obtained.

DISPOSING OF SPOIL BANKS.

Although hydraulic mine filling originated as a matter of necessity, it was developed to serve many different purposes in the anthracite industry. The huge unsightly spoil banks, the silent evidences of the waste caused by an exacting market, began to disappear and now are the exception rather than the rule.

The primary utility of hydraulic mine filling—disposal of waste—was anticipated, and from the development and expansion of the system a most commendable change in the appearance of the landscape about the collieries is being brought about. The spoil banks are being rehandled, the marketable sizes of the coal are being reclaimed, and the unmarketable product called culm is being hydraulicked into the mine workings to serve as a support in place of existing pillars when they are reclaimed. This disposal of refuse embodies also the hydraulicking of the breaker refuse directly with the bank refuse. In addition, in many instances, boiler ashes and even refuse breaker rock and “slate” from the breaker are crushed into pea and smaller sizes and hydraulicked into the old or abandoned mine workings. These methods of waste disposal appeal to landowners because they make the valuable space occupied by useless spoil banks available for colliery buildings and for habitation, and assist in solving the problem of how to provide space for needed buildings in mining districts. Several thriving mining villages are at present located on ground previously occupied by spoil banks.

LESSENING STREAM POLLUTION.

Along with the developments mentioned came the realization that river flats, after periodic floods, had been subjected to the deposit of enormous quantities of culm mixed with fine clay, loam, and silt.

* Davies, J. B., Anth. Coal Operators' Assn., February, 1908; also 1897 Rept. of the Bureau of Mines of the Department of Internal Affairs of Pennsylvania, 1898, p. XXXV.

State and local regulations were directed against such conditions, and a number of civil and criminal suits at law hastened the installation of filling systems whose operation has produced valuable results.

The extent of stream pollution and of land damage as a result of dumping mine refuse along watercourses is often underestimated, but may be realized by a casual survey along rivers with tributaries among the coal fields. Almost every spring the tributaries overflow their banks and enormous quantities* of culm and silt are washed into the rivers, which deposit the mixture on the lowlands or in the channels down stream. Not only is the transported material a source of destruction to vegetation and improvements along the bottom lands, but the action of the streams involves a great waste of natural resources as proven by the fact that during each of the years 1910 and 1911 over 90,000 tons of coal was dredged from the North Branch of Susquehanna River several miles below Wilkes-Barre, Pa.

METHODS OF HYDRAULIC MINE FILLING IN THE DIFFERENT FIELDS.

Proper credit has been accorded the middle field in which hydraulic mine filling originated. However, the process, like many others, was studied and the methods were improved by mining men in other sections of the anthracite region. Similarly, in 1893 a party of German engineers visited this country, gave special study to the subject, carried home with them the ideas in practice, and developed and expanded them along the lines well established in the Wyoming Basin, where the method of hydraulic mine filling had been applied in a practical and commercial way. In view of the above facts, it is not altogether surprising to note that the general application of the system in the northern field is considerably more advanced than in the sister fields. This is mainly due, however, to the exceptionally favorable geological conditions.

The beds are much less folded than in the other anthracite fields and large areas of the coal-mine workings are flat. In places folding necessitates chute and pitching workings, especially in the western part of the Wyoming Basin, but their proportion is so small in comparison with the general prevalence of such workings in the middle and southern fields as to be practically negligible. In many localities where filling has been practiced for a number of years, the benefits are being generally realized, and by those who have already benefited by the system it is being rapidly and efficiently extended.

* Report of the commission appointed to investigate the waste of coal mining with the view to the utilization of the waste, 1893, pp. 133-134.

As previously stated, this process originated in the middle field, but the extent of ground not underlain with coal, the unfavorable geological conditions, and the consequent greater mining cost did not at first necessitate nor make feasible the application of hydraulic mine filling to any considerable extent. In fact, hydraulic mine filling has not been extensively employed in the middle field with a view to realizing greater ultimate coal extraction, but has been adopted locally for surface support, for extinguishing mine fires, and for arresting squeezes. It can not be said that pillar coal is being reclaimed to any appreciable extent by the use of hydraulic filling, but the practice is being rapidly introduced and extended. Experience has taught the miners how to overcome the disadvantages of steep dips and the enormous pressures, from the confined water and filler, on the bulkheads. Also many of the recently planned gangways and chambers are being started with a view to possible hydraulic filling, and crosscuts and airways are being driven in such manner as to favor the future use of such filling.

The southern field, where this system was early used and where it was extensively employed for fighting mine fires, has not developed its installation for economic purposes as rapidly as have the other fields. However, its advantages and its practicable application in the middle field have caused a renewed interest in its application to the steeply inclined workings of the southern field. Especially is its application to the so-called thinner beds, ranging from 4 to 10 feet in thickness, being considered. It must be borne in mind that in this section, as well as in all other sections of the anthracite fields, the beds that were most profitable were developed first and were mined nearly to exhaustion before the thinner and impurer beds were opened and developed. The later development naturally brought improvements in mining methods and also modifications that were adapted to the new conditions. Not so very long ago, when most mining operations were conducted in the Mammoth bed, 30 to 50 feet thick, a bed only 10 feet thick was considered unminable.

MATERIALS USED OR AVAILABLE FOR HYDRAULIC MINE FILLING.

CULM.

The material first used for mine filling was that part of the mined product that passed through the smallest-sized screen used in the preparation of anthracite for market. This fine material is known as culm, a word that is said to be derived from the Welsh term "cwlwm." The material is sometimes called silt.

The entire anthracite field has been explored by operators, individual engineers, and representatives of the Bureau of Mines for

material available for filling mines by the hydraulic method. There are several extensive glacial-till deposits in the northern anthracite field,^a but in the middle and southern fields the hills are mostly composed of hard sandstone, exposed in numerous outcrops, and the covering of soil is generally thin.

The quantity of culm available for hydraulic mine filling depends on various mining conditions which are more or less peculiar to the individual fields. In mining on heavy pitches, as in the southern and middle fields, the quantity of "dirt" and refuse brought down from the coal faces is enormously increased by the crushing and sliding of the larger broken pieces. Breakage in loading mine cars, in transport, and in preparation at the breaker greatly increases the percentage of unmarketable small material. Consequently the quantity of culm is much larger in these fields than in the northern field, where the beds do not dip so steeply and where less fine stuff and impure coal is included in the coal sent to the breaker.

In the whole region the refuse impure coal, and the particles of coal that pass through the smallest-size commercial screen in the breaker are practically of slight value for fuel on account of a lack of suitable furnaces in which to burn them.

In the middle and the southern fields the greater proportion of the run-of-mine product is loaded into the mine cars from the chutes and is taken to rock chutes or directly to the breakers, where the whole product is passed over screens, washed, and separated. In the northern field the methods of mining and preparing coal are somewhat different. The coal is cleaned at the face by the miner and is again cleaned on the traveling tables and the picking tables in the breakers before it passes to the crushers to be broken into the smaller or commercial sizes. The process of breaking down this coal is performed as skillfully as possible, and the smallest possible percentage of culm is produced. The small sizes have some heating value, but unless a market is available for them they must be wasted—either run into the mines or sent to the spoil banks. However, there are few surface spoil banks at the present time, and hydraulic filling is being conducted systematically and applied wherever possible. Thus far culm alone has been the chief material used for hydraulic filling, because of uncertainties as to the source and available supply of other material, such as sand, gravel, loam, and clay, and because of the difficulties of transportation. As a result of the scientific investigations now in progress relating to the utilization of the finer sizes

^a Darton, N. H., *Sand available for filling mine workings in the Northern Anthracite Basin of Pennsylvania*: Bull. 45, Bureau of Mines, 1913, 38 pp.

for briquetting and firing under boilers by compressed-air blowers the culm may become of commercial value, so that its use for mine filling may ultimately be discontinued.

ASHES.

Although it is considered a part of the efficient operation of an anthracite steam plant to dispose of ashes by mixing them with culm from the breakers and washeries, the idea of hydraulicking this refuse into the mines is of comparatively recent date. With the thousands of tons of coal consumed daily for steam purposes about a large anthracite mine, a considerable quantity of ashes is available for use as filler. These ashes, if properly mixed with the culm and other refuse, make a filler that is easy to transport and becomes compact in the mines. Like culm, ashes will also withstand severe pressure; their compressibility is comparatively small, and their utility as roof support is unquestioned.

Near large cities considerable material for filling can be procured by the collection of ashes and the material excavated in the construction of buildings and in grading streets. All larger cities have systems of collecting ashes and garbage, and by proper arrangement the ashes collected by a city scavenger may be made available for filling mine workings under or near the city. Many industrial plants over mines, particularly over old workings, can make arrangements to hydraulic the ashes from the boilers into the mines by means of bore holes and pipes leading to predetermined points. The ashes from outlying industrial plants and from railroad engine houses might also be made available by being transported in returning empty cars.

CRUSHED BREAKER REFUSE

An appreciable supply of filling material may be made available by crushing breaker rock, "slate," and bone; also clinkers from spoil-bank fires. An effective method of disposing of the rock separated from the coal dumped into the traveling tables or picking chutes is to convey it to gyratory or swinging hammer crushers, which break it down to chestnut size and smaller and make possible its transportation into the mines through either troughs or pipes. All manner of mine rock,^a except strata between the different coal beds, can be crushed and hydraulicked into the mines.

In most heavy-pitch mining all the material is loaded into mine cars as it flows from the chutes and is dumped into the picking chutes, where the coarse refuse is taken out and sent to the spoil bank. Later the installation of crushing facilities sufficient to crush this

^a In the anthracite region the term "mine rock" includes both top and bottom rock of the coal beds and the partings of shale or "slate."

refuse and make it available for hydraulic mine filling may be practicable. Although the quantity of crushed rock so obtained would be comparatively small, the practicability of its use has not been generally tried out. However, its use no doubt will become more extensive, as it has great supporting value, so that the installation of the additional crushing facilities is undoubtedly a matter of only a short time.

As to refuse from waste-bank fires, this is available only where such fires have burned or are still burning. Experience has taught the futility of attempting to extinguish spoil-bank fires. The only effective way to overcome them is to attack the pile by hydraulic methods, flush the burned material to crushers, and later transport it in similar manner into the mine workings.

SAND, GRAVEL, CLAY, AND LOAM.

The availability of sand,^a gravel, clay, loam, and river silt is limited practically to the northern field. If the various glacial till deposits in the Wyoming and Lackawanna Valleys can be made available the problem of procuring material for hydraulic mine filling will no doubt be considerably simplified. Several attempts to utilize the deposits have been made in various sections of the two valleys and although it can not be said that the undertakings were successful, the abandonment of each of the projects was due more to lack of persistent effort and to crude methods than to the engineering impossibility of the undertaking. In considering the installation of hydraulic-filling equipment, it is necessary to consider the source of water supply as well as the availability of the material to be used.

Clear sand is used extensively as a mine filler in Upper Silesia (Germany), but has been used only to a small extent in the anthracite fields. Medium-size gravel, sand, and loam in moderate proportions make excellent filler. Clay alone does not make a good filler. Sandy loam may be used by itself, but better results are obtained if it is mixed with other material.

The most effective filler is one in which the finer constituents form a cementing bond so that after the removal of the supporting ribs of coal the material will be compact and withstand considerable pressure before sloughing to an angle of more than 45 degrees. When the angle of repose of the supporting material or filler is less than 45 degrees, it is necessary to build confines or dikes—usually called lagging—so as to retain the filler in place. The filler must also serve the purpose of reenforcing pillars and of filling small

^a See Darton, N. H., Sand available for filling mine workings in the Northern Anthracite Coal Basin of Pennsylvania: Bull. 45, Bureau of Mines, 1912, 33 pp., 8 pls.

crevices in ribs that are beginning to chip or are undergoing "air slack."

In the northern field practically all the available deposits of sand, gravel, clay, and loam are over the worked-out parts of the mines. A considerable supply of filling material may be found in the bars of rivers carrying fine culm, sand, loam, and clay in suspension and depositing those materials at points that make them available for filling if pumping equipment or other means of transportation is installed. The materials may be utilized directly at the place where they lie or at some distance from it by means of bore holes and pumps of various types; the intake of the bore hole should, of course, be considerably above the high-water stage of the river.

In the middle and the southern fields the only available material is found in hillside wash and the soil overlying the rock in the coal basins. Much of the rock is of rather soft and friable texture and might be crushed at a low cost. The material might be hydraulicked to either mine openings and crop holes or to bore holes drilled for the purpose of conducting it to predetermined points in the workings.

It is possible that eventually the price of coal and the quantity of filling material available will become so adjusted that operators will find an advantage in importing such material in coal cars that now return empty to the mines. In such event, suitable filling material will no doubt be found in practically inexhaustible quantities within short hauling distances.

GRANULATED SLAG.

Although granulated slag is used only in Europe, in the vicinity of blast furnaces, it has natural cementing qualities, is very strong, and is seemingly an ideal filling material. It has not been used in this country to any great extent, at least not in such quantity as to show how it compares in cheapness and efficiency with other materials. Along the Lehigh Valley as far east as Bethlehem and in the iron and zinc ore belts of New Jersey large quantities of slag or clinker are available.

CRUSHED ROCK.

Crushed rock and sandstone other than mine refuse has not been used to any appreciable extent, and the practicability of its use is somewhat conjectural. There are large masses of rock within comparatively short distances of many of the mines and some engineers are of the opinion that this rock can be crushed and used for filling at such a cost as to be commercially practicable. The method of transportation is a serious problem on account of the distance the material must be moved. The railroad haul is practically too short

to warrant the expense of loading and unloading under present conditions, and yet the distance that the material must be moved is, as a rule, too great to make sluicing feasible.

The cost of quarrying and crushing suitable rock is estimated at 35 to 75 cents per cubic yard. This charge makes the cost of crushed rock considerably in excess of that of other granulated material, so that it is doubtful whether this material will be much used for filling purposes for some time to come.

Attention is called to the following table* compiled from tests conducted for Eli T. Connor and William Griffith in connection with their investigations of the mining conditions under the city of Scranton, Pa. The tests were made at the Fritz engineering laboratory of Lehigh University under the direction of Prof. F. P. McKibben and W. H. Conklin, engineer in charge of the laboratory.

* Griffith, William, and Conner, E. T., Mining conditions under the city of Scranton, Pa.; report and maps: Bull. 25, Bureau of Mines, 1912 p. 55.

Results of tests of compressive strength of various forms of roof support.

No. of test	Construction tested.	Net tons per square foot required to produce compression of—						Compression and load at end of test.		Remarks.
		1 per cent.	3 per cent.	5 per cent.	10 per cent.	20 per cent.	30 per cent.	Compression.	Load.	
1	Rectangular piers of mine rock.		0.8	1.4	2.7	9.5	23	Per cent.	Tons per sq. ft.	Average construction; voids not filled. Wall constructed; voids filled with small and above stuff. Average construction.
2	Circular pier of mine rock.		3.5	5.67	11	22	38.5	31	42.5	
3	Timber crib filled with mine rock.		.6	1.37	5.11	20.3	31.5		104.5	In these tests the material was not confined, but was free to expand laterally.
4	Pile of broken sandstone; small pieces.				4	9.3	22.4	45	144	
5	Pile of small-size broken sandstone and sand.			1.6	4	13.5	35	45	75	
6	Pile of broken sandstone; large pieces.		3.6	5	9.1	28.4	.37	27	56	
7	Pile of Coal-Measures sandstone similar to No. 6.	0.8	2.1	3.5	9	33.4		63	108	In these tests the material was confined and could not expand laterally.
8	Pile of river sand.							35	666	
9	Broken sandstone in cylinder.		3.33	5.55	13.32	46.6	98.6	23	666	
10	Broken sandstone and sand in cylinder.		3.5	5.77	24.42	308.5	25	51	666	
11	Dry coal ashes flushed in with water.		1	1.86	5.32	10.8	22	33	666	This test was made at the Dickson Works of the Allis-Chalmers Co. in Scranton, by William Griffith.
12	Wet culm flushed in cylinder; partly dried (average of two tests).	2.44	8.9	14.28	35.52	138.7	444		666	
13	Dry sand in cylinder.	.88	3	5.27	33.3	129	499	32.2	666	
14	Wet sand flushed in and partly dried.	8.4	39.3	67	173.8	555.4		20.75	666	
15	Concrete made of cement, sand, and gravel; 4 months old; 1 barrel Portland cement to each cubic yard of concrete (about one part cement to seven parts sand and gravel); piers 3 inches by 2.81 inches by 3.85 inches high.	9	84					Gradually crushed to pieces under continuous load of 45 tons		
		Cracked.								

TRANSPORTATION OF FILLER.

SURFACE TRANSPORTATION.

The breaker or washery by-product that is hydraulicked into the mines is that which passes through the finest screens used in preparing the commercial sizes of anthracite. The openings in these screens are punched and vary from three thirty-seconds to three sixty-fourths of an inch in diameter. To the washery product the waste from the "chippings" elevator or scraper pits is added; by means of a trough the mixture is conveyed to a $\frac{1}{2}$ -inch screen, and what passes through enters the bore hole or the receiving pit. Throughout the Wyoming and Lackawanna Valleys, where the surface slopes are gentle, many of the colliery yards are drained to this same pit, which thus receives the excess storm water.

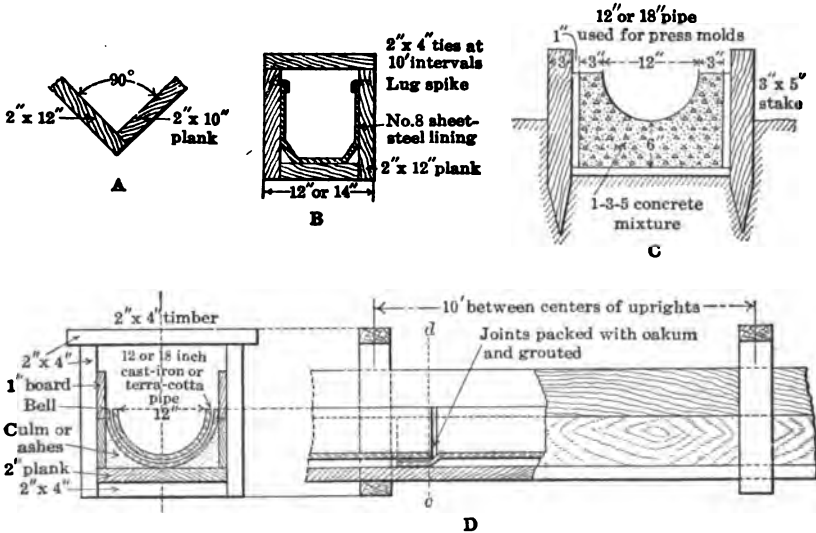


FIGURE 1.—Types of troughs used for surface transportation of mine filler. A, plank trough; B, plank trough with sheet-iron lining; C, concrete trough; D, cross-section on line c-d, and longitudinal section of terra-cotta or cast-iron trough.

TROUGHES.

Where hydraulic mine filling has not yet been introduced the refuse mentioned is collected in a hopper and, by means of gravity lines, pumps, or elevator and scraper lines, is conveyed into troughs or cars to be delivered to the spoil or waste bank. The troughs are constructed of (1) plank (fig. 1, A) or plank with a sheet-iron lining (fig. 1, B and D), (2) terra-cotta or cast iron (fig. 1, D), or concrete (fig. 1, C).

Sheet-iron lined plank troughs.—The sheet-iron lining is generally bent so as to form a semicircle (fig. 1, D) or a half hexagon

(fig. 1, B), and is placed in the plank trough in lapping or flowing joints. Such sheet-iron lining is used only where the inclination of the troughs is less than one-half inch to the foot.

Terra-cotta troughs.—Either round or half-round terra cotta is also used and may be held in place by either boards or concrete. When held in place by boards, a rectangular form is constructed and is partly filled with culm or ashes, so as to form a bedding for the terra cotta (fig. 1, D). Oakum and cement mortar are used to form the joints.

Concrete troughs.—Concrete troughs (fig. 1, C) are constructed in various forms and sizes. The concrete is made of 1 part cement, 3 parts sand, and 5 parts crushed stone. Broken "slate" and rock from the breaker make a satisfactory substitute for the crushed stone, as their resistance to wear is very nearly that of the matrix. The concrete is molded by the use of either a V-shaped wooden trough or a round pipe embedded in it and removed before the final set begins. Whenever large troughs requiring large quantities of concrete are built, large pieces of uncrushed mine rock are embedded in the concrete.

MECHANICAL CONVEYERS.

The mechanical devices in use for the transportation of material on the surface consist of (1) elevators, (2) scraper lines, (3) pumps, and (4) dump cars. In addition, gravity planes may be used where surface features are favorable, and aerial tramways where the surface is rough and broken.

Elevators.—The elevators consist of ordinary sprocket links to which at intervals of about 6 feet are fastened perforated buckets large enough to handle the material as it is deposited in a hopper at the foot of the elevator line. The plane of the elevator is inclined at an angle of about 70 degrees. The length depends entirely upon the height required to furnish sufficient grade for the material to flow through sheet-iron pipes or terra-cotta troughs to the spoil bank, to the bore holes, or to cars.

Scraper lines.—Scraper lines are used to convey the waste culm or refuse over a considerable distance from a common collecting point to the spoil banks or to bore holes. The distance as a rule does not exceed 1,000 feet. Scraper lines are intended mainly for horizontal transportation, whereas elevators are used for vertical transfer. At many mines the distance from the source of the refuse used to the point of entrance to the mine is too great for scraper-line transportation, and pumps are used.

Pumps.—The ordinary piston or centrifugal pump has been found to give the best service in handling the refuse if the material is mixed with proper proportions of water (seldom in less proportion than 10 parts water to 1 part filler). It is good practice to use pipe of the

smallest practicable diameter, thus avoiding the possibility of cross currents or eddies, which are apt to cause frequent blocking of the pipe. Cast-iron pipe, with either bell-and-spigot or flange joints, gives excellent service. After installation the pipe should be turned at stated intervals so as to obtain the maximum wear of the full inside surface.

Dump cars.—Dump cars run on small spurs of narrow-gage track are employed to bring ashes from the boiler plant, which is generally at some distance from the breakers, to the point of common collection, either at the breaker or at the bore hole. Both steam and compressed air have been used to force boiler ashes or breaker refuse through short lines of pipe and have proved effective for the purpose.

INTERMEDIATE TRANSPORTATION.

Intermediate transportation is accomplished through shafts, bore holes, slopes, crop falls, and cave holes.

SHAFTS.

A shaft may be used to contain a pipe line that runs to the lower workings; or if the shaft is not used for hoisting or ventilating purposes, the filler may be run into the shaft, no pipe being employed, and may be allowed to find its way by natural flow into the lower parts of the workings. The use of the shaft without a pipe line is unsystematic; and unless the shaft is shallow, the bed rather steeply inclined, and the workings not very extensive, the practice should be discouraged.

The modern and more systematic method of procedure is to place a pipe line in the shaft (fig. 2, A). This is joined by proper connections (fig. 2, E) to an interior pipe line, through which the material is conveyed to any desired point. The pipe generally is made of wood, terra cotta, steel, wrought iron, or cast iron. The shaft line is securely fastened to the shaft timbering by means of clamps and bolts, so that repairs or necessary changes can readily be made.

Care is taken to have at frequent intervals proper connections to make possible the taking apart of the line without suspending its entire length. The clamps are so constructed that they snugly fit the outside of the pipe and are placed so as not to interfere with taking apart or assembling the various forms of joints. It may not be necessary to place special timbers in any shaft, so that the clamps may be bolted directly to the buntons or wall plates (fig. 2, B, C, and D).

BORE HOLES.

Bore-hole transportation is the method most commonly used, but it involves several important details. The first consideration is the

selection of a practicable location. At many mines the bore hole can be driven to the point most desirable in the mine, and connection with the bore hole at that point may be established by driving a special opening to receive the line from the surface. The surface location of the top of the bore hole is usually fixed by reference to the topography of the ground about the breaker or by some particular geological feature of the mine workings.

USE OF LINING PIPE.

The bore holes are drilled by means of drilling rigs of standard reciprocating type, the method of drilling being in general similar

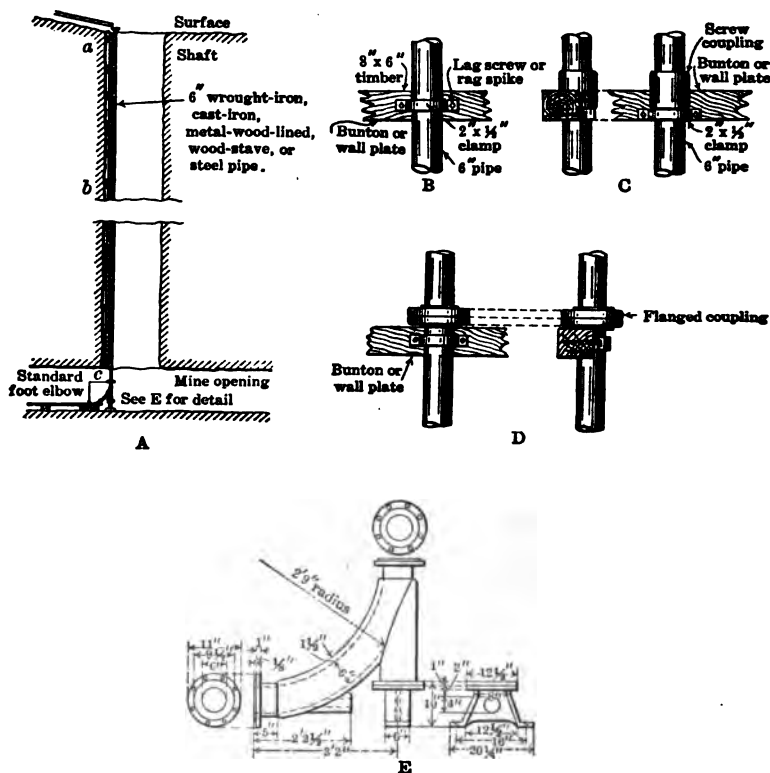


FIGURE 2.—Methods of pipe arrangement in intermediate transportation. A, sectional view of pipe line in shaft; B, detail of clamping at a; C, detail of clamping between couplings at b, front and side elevations; D, detail of clamping at couplings at b, front and side elevations; E, detail of foot elbow for filling-pipe standard at c.

to that used in drilling wells into hard rock. The diameter is fixed by the quantity of filler that is to be handled. In order to preserve and maintain the hole, the drive pipe, from surface to hard rock, is protected by means of a lining pipe (fig. 3, B, C, and D), which may be of metal or specially vitrified terra-cotta. Metal lining pipe should be nonmalleable and so placed that it can easily be broken into small

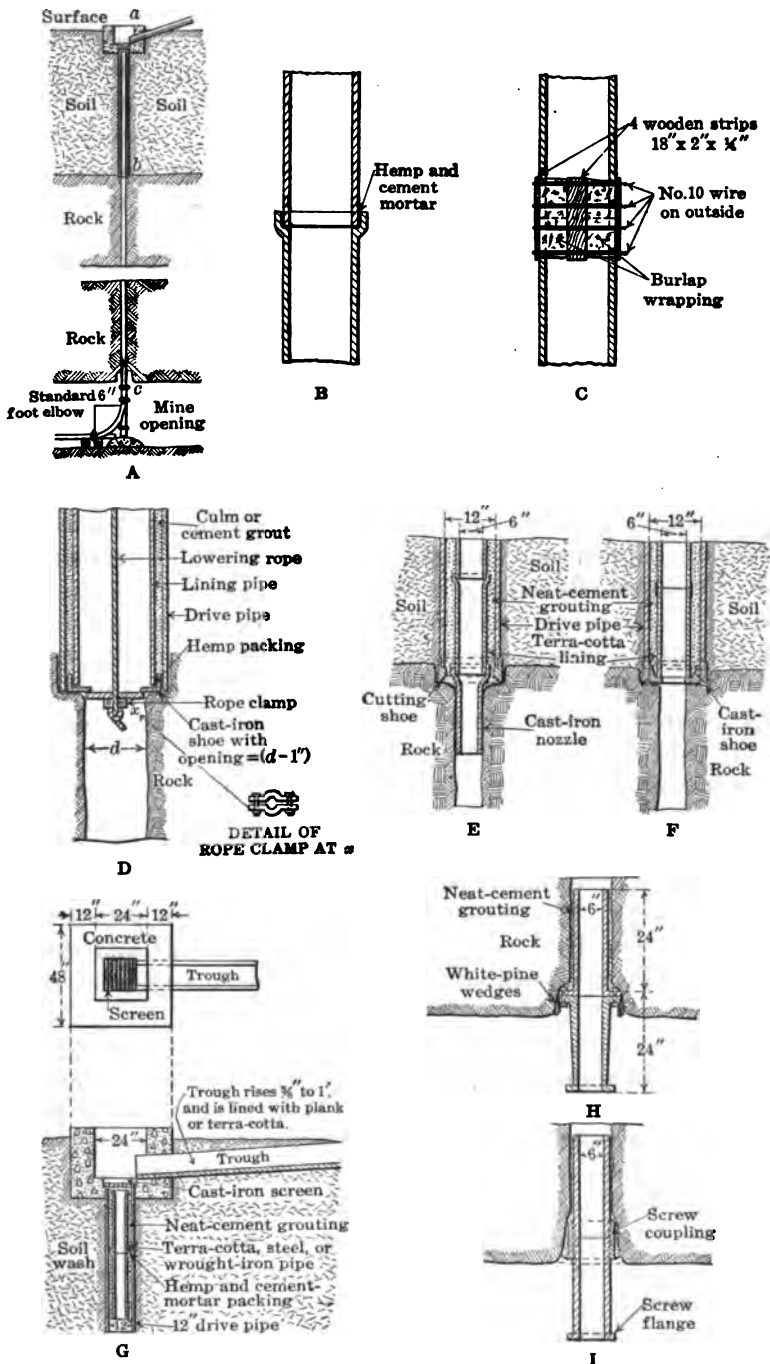


FIGURE 3.—Methods of pipe arrangement in intermediate transportation of filler. A, general cross-sectional view; B, terra-cotta bore-hole lining with bell-and-spigot joint; C, straight cylindrical metal or terra-cotta bore-hole lining; D, method of placing bore-hole lining; E, detail at b, showing arrangement of rock nozzle; F, detail at b, showing alternate arrangement of rock nozzle; G, detail of arrangement of parts at a; H, detail of arrangement of parts at c; I, detail of alternate arrangement of parts at c.

pieces and removed if it wears out. It may be constructed with special lock flanges or with ordinary bell-and-spigot ends. On account of the large diameter of hole required for standard flange joints, such joints are impracticable. Ordinary bell-and-spigot joints are packed with oakum and cement mortar as the pipe is lowered into the hole. The terra-cotta lining is either of the plain cylindrical type or of the bell-and-spigot type.

In using the plain cylindrical terra-cotta pipe the lengths are joined by means of a triple thickness of burlap wrapping extending about 9 inches on either side of a joint. The wrapping is stiffened with 2 by 4 by 18 inch wooden strips firmly held in place by bands of No. 10 galvanized wire. The three types of lining described above require rock nozzles (fig. 3, E and F).

The method of inserting the pipe in the bore hole is briefly as follows: From a tripod, or the rocker arm of the drilling machine, a single-sheave pulley is suspended. Through the pulley is passed a hemp rope of sufficient strength to support the full length of lining pipe. To the end of the rope a clamp is attached above a knot in the rope. The clamp supports a circular plate of a diameter 1 inch less than that of the hole in the rock. This plate supports the cast-iron rock nozzle or shoe, which in turn supports the lining pipe. About 3 feet above the ground another clamp is placed on the rope. The rope is paid out on the ground the full length. Two lengths of terra-cotta lining pipe are passed over the rope and the pulley is hung from the tripod or the walking beam of the drilling machine, care having been taken to pass the pulley through the lengths of pipe so that its position is between the bottom clamp and the loose end of the rope. The loose end of the rope is then passed around the drum of the drilling engine three or four turns and the slack taken up. This brings the lining pipe in a vertical position over the hole.

The joints are then packed with oakum and cement mortar, and the pipe is lowered into the hole until its upper end is nearly flush with the top of the drive pipe. The rope is clamped and held in position by the lower attachment, the block is taken down, the rope is paid out on the ground as previously, and two more lengths of pipe are strung on. The block is again attached to its hangings, the slack of the rope is taken up by means of three or four turns on the engine drum, the whole string is lifted so that the clamp can be removed and the additional pipe connected to the two lengths already placed, and the joints are packed, the pipe lowered into the hole, and the rope clamped as before. This process is repeated until the entire lining has been inserted. The space between the outside of the lining pipe and the inside of the drive pipe is then filled with cement grouting.

When ordinary screw-sleeve gas-pipe lining is used, it is lowered into the rock for several feet in order to insure perfect discharge and the protection of the rock joint and the drive pipe. Such lining pipe is suspended by means of clamps over the receiving basin, usually of concrete, and is protected by means of a screen similar to that shown in figure 3, G. At the bottom or foot of the bore hole, either a special cast connecting piece, as shown in figure 2, E, or figure 3, H, or a short piece of wrought-iron pipe fitted with a screw flange is inserted up the hole for 2 or 3 feet (fig. 3, I). The space between the outside of the connecting piece and the wall of the bore hole, after having been packed with hemp, wedges, or burlap, is filled with a grouting mixture of proper consistency.

BLOCKING OF BORE HOLES.

A serious source of trouble, and in many instances of prolonged delays, has been carelessness on the part of the person in charge of the intermediate transportation, usually manifested either by poorly regulated water supply or by large pieces of filler being allowed to pass the top screen. If through such carelessness the hole is blocked, reopening may entail considerable expense. To lessen the delay it has been found expedient to have ready for insertion at any time a water pipe an inch or an inch and a half in diameter with a steel point and a series of holes extending 2 or 3 feet above the point. When the bore hole becomes blocked, the foot elbow is removed, the pointed pipe is inserted, and water under high pressure is turned into it. The water thoroughly saturates the filler and causes it to pass off at the bottom.

Another method of reopening is to attach short lengths of 1-inch to 2-inch pipe, connect these to a mine column line, and force water up into the hole, thus washing out the material. In case it is impractical to arrange for reopening by water jet from below, the use of a long pointed iron rod, about 2 to 3 inches in diameter, raised and dropped from the surface by means of a windlass, has sometimes been found effective, but in general this is bad practice and should be discouraged. Where the influx of filler and water is irregular it may be expedient to turn a small supply of water, say a 1½-inch pipe running full, from an independent source into the receiving hopper at the surface.

RELINING BORE HOLES.

It frequently becomes necessary to change the lining of the bore hole because of unequal erosion; the drive pipe is sometimes destroyed in sections, and surface water and sand or gravel flow into the hole and cause serious expense or the blocking of the hole. In many such cases it has been necessary to drive a pipe over the origi-

nal drive pipe and remove the damaged drive pipe by means of a chopping bit. This procedure is, of course, very slow and expensive; and if the mine is being damaged by water or sand flowing in through the abandoned bore hole, it is cheaper to drill a new hole and fill the old one with concrete. If, however, the lining only is affected, which may be readily discovered by observing the nature of the filler deposited, the operation of reopening is simple if the lining consists of earthen pipe or nonmalleable iron pipe. An ordinary drilling rig and a chopping bit are employed, and all the broken material, such as lining and cement grouting, is lifted from the hole with a sand pump. A new lining, with its accessories, is then placed in the same manner as was the original lining.

If a loose, uncemented metal lining pipe is used, it may readily be withdrawn from the hole by means of tripod, block and tackle, and windlass. It is, however, highly important that the practicability of having duplicate holes be kept in mind. A second hole should be provided wherever possible, and in order to insure the proper operation of both holes the filling material should be run through each on alternate days.

With few exceptions it is more economical to drill a new bore hole when the lining wears out. This is particularly true if the metal lining extends through the rock and into the mine, as was the old custom, or if the lining is grouted.

SLOPES.

If the hydraulic filling is to enter the mine by means of a slope, a pipe is placed with appropriate facilities for anchorage. Anchorage may be accomplished by the use of mud sills (*a*, fig. 4) to which the pipes are fastened by means of clamps and rag spikes, or by rag-spiking the clamps directly to props set for this purpose (*b*, fig. 4), also by suspending the pipe line by a chain with one lap around the pipe and fastened to the roof or the props. On account of the great expense entailed in placing the pipe, it is economical to use for this purpose wood-lined pipe in sections having cast-iron flanges held together with bolts, eight or more in number, depending on the pressure in the line.

CAVE HOLES.

In places the surface subsides very slowly over old abandoned mine workings, especially where the beds are thick and the pillars left have been too small; eventually over such mine workings cave holes appear which are often a nuisance as well as a possible cause of injury to unsuspecting trespassers. In order to close such unsightly and dangerous holes, filling material is delivered to them and allowed

to enter the workings and fill as many of the openings as the inclination of the workings and the passageways available will permit. When the cave holes are filled to the former level of the ground, the danger to either life or property is of course removed.

UNDERGROUND TRANSPORTATION.

The method of conducting filler from the receiving point to the openings to be filled requires careful study and intimate knowledge

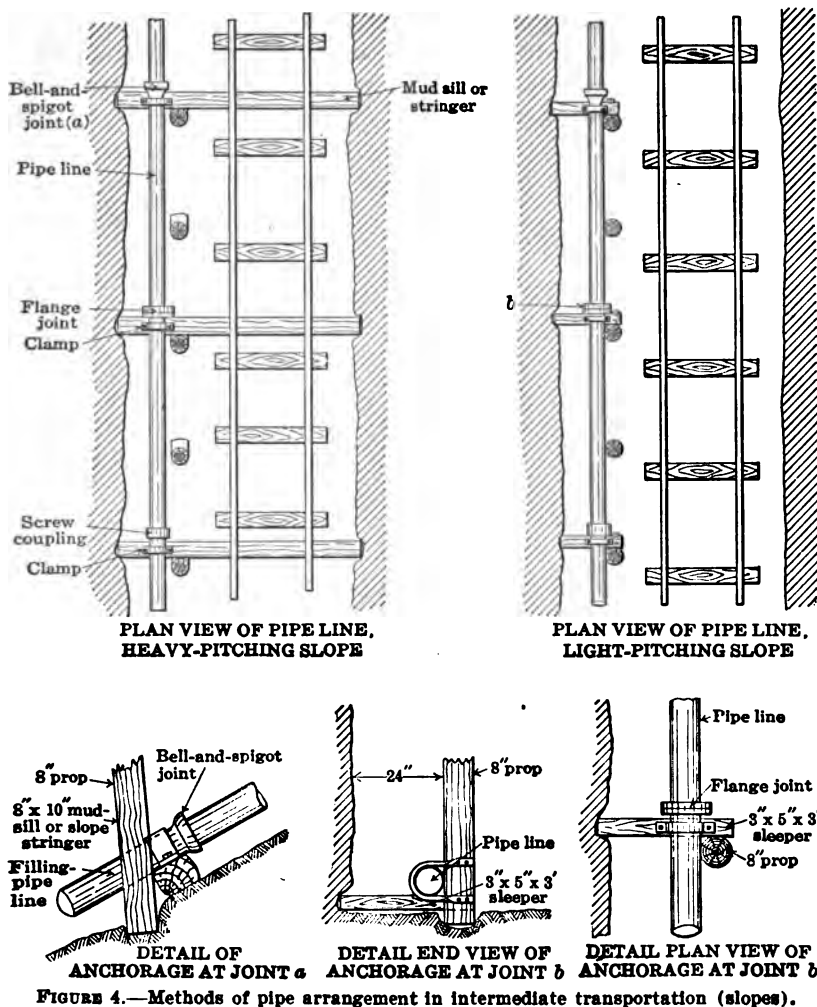


FIGURE 4.—Methods of pipe arrangement in intermediate transportation (slopes).

of the physical features of the mine. It is obvious that a flat mine offers considerably greater difficulties than one working moderately inclined beds and much less than one working heavy-pitching beds. The various methods that may be used, from the cheapest, although

probably not the most economical, to the most systematic and therefore the most expensive method, are here presented.

UNCONFINED FLOW.

When the filling material is discharged into the mine freely, as from a shaft or bore hole, there must be sufficient inclination below the point of discharge to cause a current that will hold the material in suspension. Consequently, in horizontal or flat beds the moving of filler by unconfined flow is impracticable. In beds with an inclination greater than 10° transportation by unconfined flow is practicable if the material is not to be delivered at too great a distance from the point of influx. By some care in planning, or under

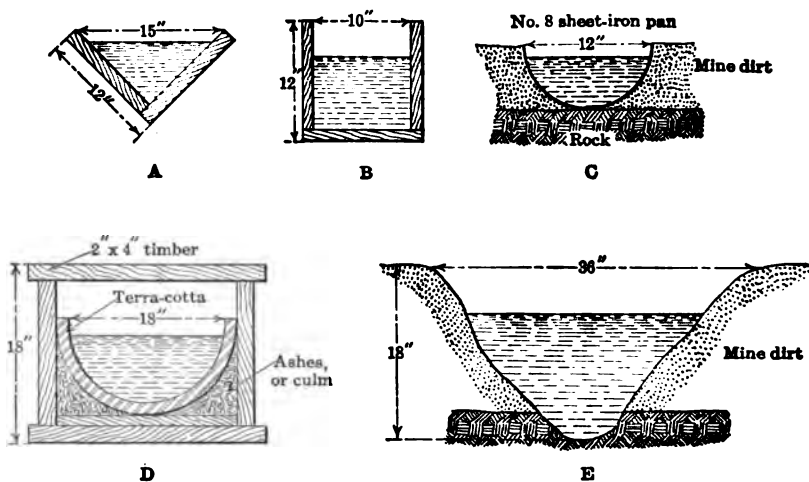


FIGURE 5.—Types of troughs and channel for underground transportation.

the supervision of a careful miner, a dirt ditch or channel (fig. 5, E) may be washed out and the filler conducted satisfactorily into the openings to be filled. Great pains, however, should be taken to maintain the channel in such condition that it will not become blocked or dammed, thus avoiding a source of considerable expense and damage. Such damming renders filling by unconfined flow dangerous when practiced in sections of higher elevation than those in which men are employed. In one instance a dam was formed by a partial obstruction in the channel. The slow seepage of the bearing water permitted a large quantity of material to accumulate, and eventually the accumulation started and rushed down the lower workings, into a section that was to be filled, with such force as to destroy the bulk-heads and fill up traveling ways and air courses.

TROUGHES.

Where the inclination of the coal bed is between 5° and 10° , troughs lined with sheet iron (fig. 5, C), terra-cotta (fig. 5, D), or even unlined (fig. 5, A and B), are used to convey the filler from the bore hole or shaft to the opening prepared for it. A considerable saving in the construction of troughs can be effected by the use of sheet-iron pans (fig. 5, C). The sheet iron is bent in such a manner as to form a semicircular or semihexagonal section, and by placing the sheet-iron lengths in lapping joints the filler can be conducted directly to the desired point, the cost of constructing and removing wooden troughs being thus saved.

PIPE LINES.

In horizontal or ascending workings it is necessary to employ pipes of wood, steel, wrought iron, cast iron, terra-cotta, porcelain-lined, or glass.

WOOD-STAVE PIPES.

Wood-stave pipes are probably the type most extensively used. The staves, of the regulation tongue-and-groove type (fig. 6, A), are made of hard wood, such as oak, ash, or maple, and form a pipe that is economical and highly serviceable. It is spirally wound with galvanized wire or steel bands and is given a protective coating of tar, sprinkled with "granolithic" sand, slag, or sawdust. The wire winding proves most serviceable, as the area subject to the attack of acid water is minimized, yet the wire forms a strong protection. Connection of the lengths of pipe is made by the usual male and female ends (fig. 6, B). This form of connection does not permit easy and relatively cheap cleaning of a line in case of blockage, nor does it permit the introduction of elbows, bends, tees, or valves. To overcome this objection cast-iron or wrought-iron flange connections are inserted into the line (fig. 6, C). These necessitate the use of short pieces of pipe with two male ends (fig. 6, D), as the cast-iron or wrought-iron flange connections are usually constructed for male connections. The flange connections are easily disconnected, so that in case of blockage the lengths of pipe can be rolled out of line and cleaned. One of the greatest advantages of the flange-connecting pieces is that they permit easy and inexpensive turning, so that the maximum service is obtained from the pipe.

STEEL AND WROUGHT-IRON PIPES.

Where the working pressure or water hammer on a pipe line is liable to exceed 100 pounds per square inch, steel, cast-iron, or wrought-iron pipes are used. Connections are effected by either screw or flange joints (fig. 7, A and C) or by plain sleeve couplings (fig. 7, B). The sleeve of the plain sleeve coupling is packed with oakum,

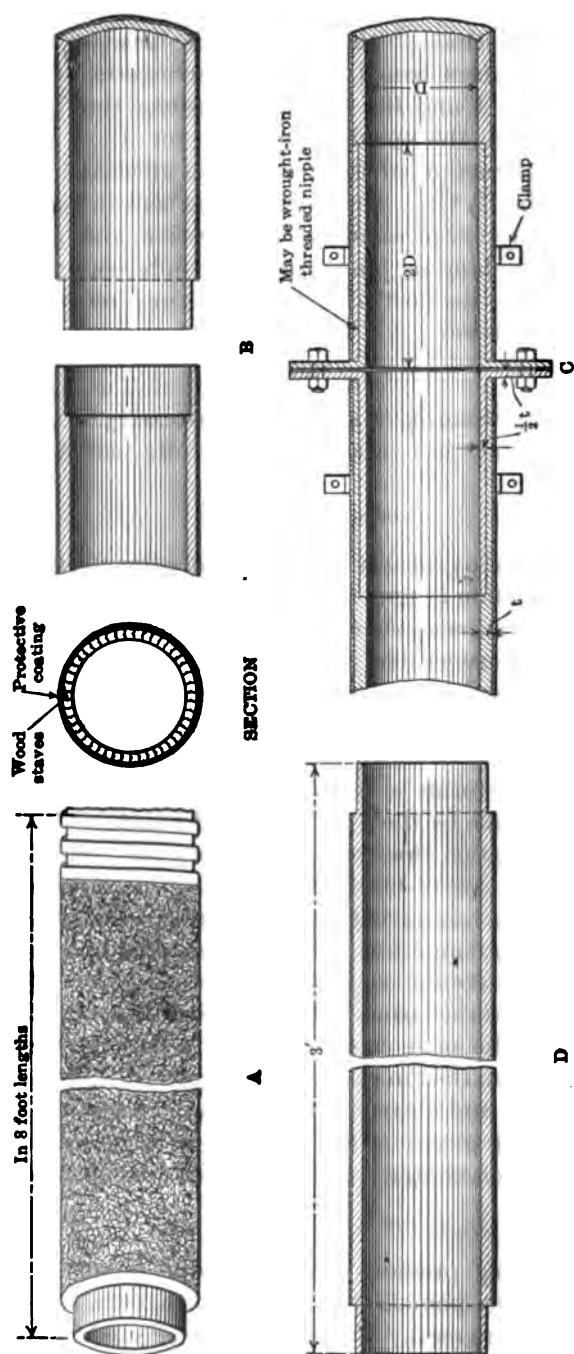


FIGURE 6.—Construction and connections of wood-stave pipe. A, regulation 8-foot length of wood-stave pipe, showing metal-band wrapping with tar and sawdust coating; B, male and female ends; C, standard cast-iron flange coupling; D, short-length pipe with male ends for use with wrought-iron flange coupling.

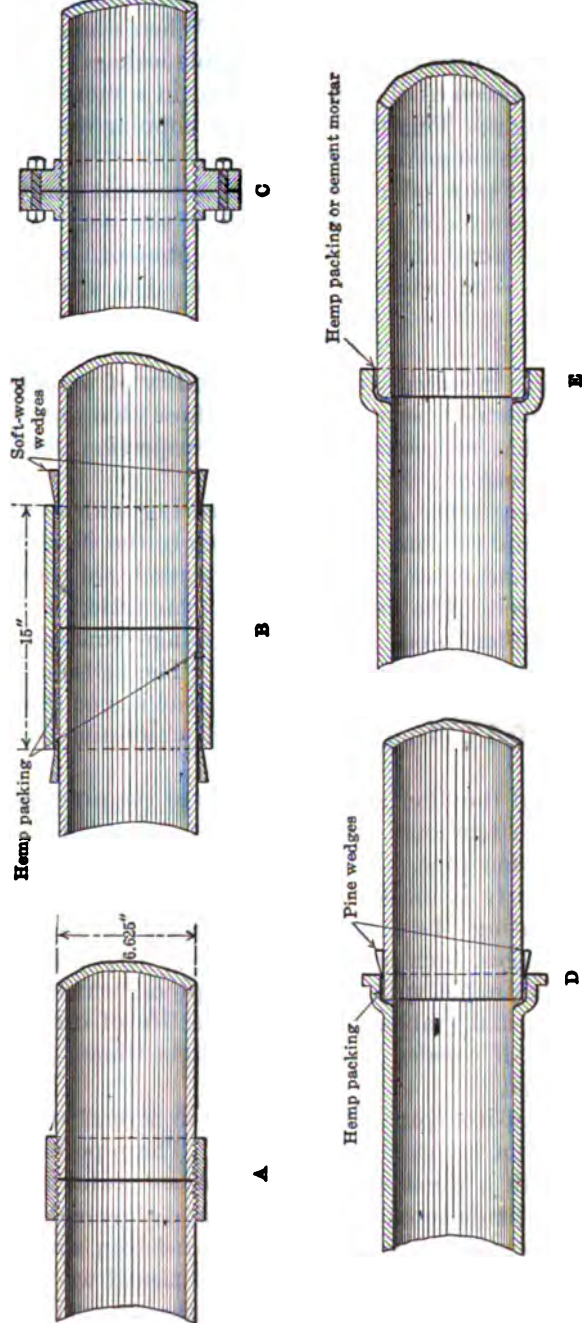


FIGURE 7.—Connections of metal and terra-cotta pipes. A, screw coupling; B, plain sleeve; C, standard flange coupling, 6 bolts; D, cast-iron bell-and-spigot connection; E, terra-cotta pipe connection.

burlap, or strands of old or discarded hemp rope, and white-pine, hemlock, or other soft-wood wedges are driven sufficiently close to each other to make the joint water tight. When water and filler are first turned into the pipe there is considerable leakage, but the oakum or other packing becomes filled with silt and in a short time the joint becomes water tight. This type of pipe is popular on account of its low price and wide range of application.

The flange-joint pipe offers advantages somewhat similar to those of the plain pipe except that flange joints are not as easily kept tight and are not adaptable to irregular alignment. Obviously, the cost of such pipe is considerably greater than that of the plain pipe.

CAST-IRON PIPES.

Where lines are installed with the probability of remaining in one place for a considerable period, especially those along traveling ways or air courses that are constantly under inspection, cast-iron pipe (fig. 7, D) is extensively used.

This pipe possesses the advantage of wearing with a continuously smooth surface and offering little resistance to the flow of the filler. The types of connections most extensively used are the flange and the bell-and-spigot. The bell-and-spigot connection is used more frequently for filler pipe, whereas the flange connection is used almost exclusively for pump-column lines. The bell-and-spigot connection is made tight by the use of oakum packing or short pieces of old hemp rope, and is firmly secured with white-pine or other soft-wood wedges.

The abrasion of the filler keeps the inside of the pipe smooth at all times, so that the frictional resistance, it is claimed, is considerably less than in wooden pipe. The feature of cast-iron pipe that gives it wide adaptability is its being made in 12-foot lengths, which, although necessitating the expense of frequent connections, afford physical advantages, such as the possibility of irregular alignment, that are not obtainable with other metal pipes, which are made in longer lengths, usually about 20-foot. The life of cast-iron pipe is about twice that of wooden pipe and about six times that of steel or wrought-iron pipe.

Wood-lined cast-iron pipe has been used in several mines, particularly on slopes, but its use is not common. The pipe has a cast-iron flanged shell with a wood-stave lining. When water is introduced, the wood lining tends to expand or swell and thus make a firm contact between it and the inner wall of the cast-iron shell. The pipe possesses the wearing and acid-resisting qualities of wood pipe and of ordinary cast-iron pipe, but by reason of the expensive outer shell and the rather short life of the wood lining, unless it is carefully and

wearing out, so that the general application of this form of pipe can not be said to be probable.

TERRA-COTTA PIPES.

In places where there is slight danger of damage from falls of roof or other external sources of injury the use of terra-cotta pipe (fig. 7, E) may be safely and economically adopted, especially if there is no back pressure on the line. The expense of frequent joints should, however, be avoided if practicable. The pipes are manufactured in 3-foot to 4-foot lengths. Ordinarily, the joints are packed with hemp or oakum with a thin layer of cement mortar as surface packing. The cement mortar is held in place until firmly set by means of a wrought-iron clamp bolted flush with the edge of the bell flange. Terra-cotta pipe, however, offers considerable frictional resistance on account of the rapid abrasion caused by the filler which soon wears off the smooth glaze. Especial care must be exercised and close supervision enforced so that the pipe will be turned in sections at proper intervals to obtain the maximum wear. The pipe can be advantageously used on all descending lines in flat workings where the filler is to be conveyed directly to the bulkheads.

PORCELAIN-LINED PIPES.

Porcelain-lined pipe has been extensively used with excellent results in European plants, although in some mines it has not been so successful and has been replaced by plain metal pipe. Owing to its high price in this country and to the difficulties of manufacture the cost of this type of pipe is so great as to make its use impracticable for hydraulic mine filling in the United States under present conditions.

GLASS PIPES.

In order to avoid the heavy expense of renewing elbows and other bends in filler lines, a remedy was sought in the use of glass pipe, particularly for bends of 45° to 90°. The experiment, however, has not proven an entire success, and the use of glass pipe will probably be desirable only under special conditions.

COMPARATIVE EFFICIENCY OF VARIOUS KINDS OF PIPES.

It must be borne in mind that wherever pipes or bore holes are used for the transportation of filler the filling must be conducted with judgment. Under no circumstances should filler be turned into a bore hole or pipe line until considerable water has been run through to remove any pockets of sediment that may have formed. Water should also be run through after the flow of filler has been turned

off at the completion of the day's work, so that the material will be completely flushed out.

In stating the efficiency of various kinds of pipes employed for hydraulic mine filling, it must be borne in mind that accurate experimental data are not available. However, the observations made below are believed to be warranted. Proper regard must, of course, be had for the geological conditions under which any of the methods is operating. For example, where the grades are undulating it is absolutely necessary to have a stout metal pipe unless the back pressure is comparatively light. Roughly, such pipe should be used when there is a difference of 100 feet between the highest and lowest points in the line. At no point throughout the system should the velocity be less than 6 to 8 feet per second. This will insure against deposition of the filler in the line.

A brief discussion of the merits of the different kinds of pipes mentioned above follows.

WOOD PIPE.

ADVANTAGES.

Wood pipe possesses a number of distinctive advantages. (1) It is immune from chemical attack; (2) it has a tough wearing surface; (3) it is easily turned; (4) it is easily cleaned; (5) its cost of installation is low; (6) it can be used under difficult physical conditions.

Immunity from chemical attack.—The chemical attack on pipe used in hydraulic filling is of great importance, especially where ashes or other acidulous filler is employed. This attack is exerted not only on the filler pipe itself, but on pumps and drainage columns as well.

Tough wearing surface.—Wood pipe has a tough wearing surface, and although somewhat less durable under steady use than metal pipe of equal thickness, it is not impaired by rusting, as is metal pipe when the line is unused for some time. Wood pipe will wear much longer if it is frequently turned slightly, say one-eighth to one-fourth turn at a time.

Ease of turning.—Being in short lengths, wood pipe can be turned with less expense than metal pipe, especially if cast-iron flange connections are placed in the line at rather frequent intervals, say every 30 feet.

Ease of cleaning.—Because of the flange connections and the short lengths of the pipe, blockages are cleaned and pipes reopened at practically minimum expense, especially where the material has to be withdrawn by scrapers or punching rods.

Low cost of installation.—The lightness of wood pipe permits easy and inexpensive handling and the cost of placing or installing is therefore reduced to a minimum.

Adaptability to difficult physical conditions.—Wood pipe can be installed with comparative ease under difficult conditions, such, for instance, as are found in old and caved workings. The lengths can be connected so as to allow considerable deflection in vertical and horizontal alignment.

DISADVANTAGES.

The disadvantages attendant on the use of wood pipe are: (1) It dries out and collapses when not regularly in use; (2) it wears unevenly; (3) it easily springs out of line; and (4) its life is comparatively short.

STEEL AND WROUGHT-IRON PIPE.

All exposed surfaces of steel or wrought-iron pipes are subject to corrosion. The effect is serious and should receive careful attention when a filling system is being planned. The wearing surface of metal pipes, although smooth, is subject to strong abrasion from the rapid cross currents and eddies in the flow. The point at which the greatest wear will occur can not be definitely fixed as the wear varies greatly. No particular precaution can be followed other than turning at frequent intervals. Where screw thimbles or screw joints are used, turning becomes rather complicated and requires a number of men, whereas if the slip-sleeve or flange connections are used the expense is considerably reduced.

Screw joints do not permit pronounced variation in alignment unless, before the pipe is placed, it is bent to conform with the alignment so that the use of flange connections becomes necessary. If a line is "sprung" around curves the outer wearing surfaces are subject to considerable strain in addition to the centrifugal force in the current and leaks and failures of the pipe result. Steel or wrought-iron pipe is more difficult and expensive to clean than is wood pipe on account of the lengths being longer and also less easy to remove unless flange connections are used.

One of the greatest disadvantages of this type of pipe is its tendency to corrode. Each morning considerable care must be observed when the mixture of water and filler is turned into the line as otherwise blockage from friction caused by the corrosion overnight may be the first thing to occur in a day's operation.

CAST-IRON PIPE.

Although there is a serious objection to the weight of cast-iron pipe of standard length (12 feet) this pipe has become rather popular for use in hydraulic mine filling. The bell-and-spigot connection allows great flexibility in vertical and horizontal alignment and because the

pipe keeps a smooth surface during use the proportion of water and the velocity of the current can be reduced to a minimum. On account of the short lengths in which cast-iron pipe is made it can be installed in caved and tortuous openings. The wearing surface must be thoroughly washed twice a day, before and after filler has been introduced, in order to guard against frequent and expensive blockages. Cast-iron pipe is easily turned if either the bell-and-spigot or the flange type of connections is used.

It should be borne in mind that all pipes when installed will sooner or later have to be removed for cleaning and should be so placed that this can be accomplished at the least expense in labor and material. When bell-and-spigot connections are used, it is advisable to insert at intervals of 48 to 80 feet flange connections so that sections of pipe can be rolled out of line and cleaned or new sections put in.

Cast-iron pipe is subject to the corrosive effect of acid water, and although it deteriorates less rapidly than pipe made of wrought iron or steel, nevertheless such water has a very destructive effect and shortens the life of the pipe decidedly. It is claimed that under a pressure of 100 pounds to the square inch cast-iron pipe will wear three times as long as wrought-iron and about five times as long as steel pipe.

WOOD-LINED PIPE.

Wood-lined pipe is used only in special cases, as in shafts or slopes. In some mines, especially where the mine water is highly acid, it has been expedient to use the filling line as a water column during the night, the main trunk line being suitably connected to the mine pump. However, on account of the great expense and the rapid wear of the wood lining, this practice now is practically discontinued in favor of the more effective plan of having separate lines, usually of cast iron, with bell-and-spigot connections, for the filling pipe and flanged wood-lined pipe for the water column.

TERRA-COTTA PIPE.

The use of terra-cotta pipe for underground transportation is rather restricted. The cost of this pipe is comparatively low, but it lasts only a comparatively short time. On account of the texture of the body of the pipe, as soon as the inner glazing is cut through erosion is rapid, and of course there is considerable resistance to the flow of filling material.

GLASS PIPE.

Glass pipe has been tried only as an experiment in the hope of obtaining a good wearing material of low frictional resistance that might be used particularly where there are changes in alignment, as

use was expected to eliminate the serious trouble experienced in metal pipe lines at a point some distance beyond a curve or bend. Pipe frequently wears beyond the bend or at the joint next beyond the bend. Consequently, glass elbows have been tried that are of special thickness just beyond the bend. These elbows wear longer, but the cost of glass pipe practically prohibits its general use, even for elbows.

PORCELAIN-LINED PIPE.

Porcelain-lined metal pipe has been used extensively in Europe, but on account of its great cost in this country its application has been impracticable. The metal shell is usually of wrought iron or steel, and the lengths are provided with flange connections. The porcelain lining is about one-half inch thick and is divided into sections about 10 or 15 inches long. The space between the metal shell and the porcelain lining is filled with a cement grouting. Eight bolts are generally used to bind together the flange joints.

The wearing qualities of porcelain-lined pipe are considerably greater than those of plain metal pipe, and the frictional resistance is considerably lower. However, when coarse material is transported the necessarily high velocity of the current in the pipe line causes the larger material to impinge on the walls of the pipe; the continual impact tends to crack and break the porcelain lining and thereby destroy its efficiency.

USE AND CHARACTER OF BULKHEADS.

The construction of the bulkheads to be placed to retain the filler in mine workings should be given careful consideration. The conditions in the workings to be filled should be carefully analyzed with a view to determining the most economical and at the same time the most efficient type of bulkhead.

TERMS USED IN ANTHRACITE FIELDS.

In selecting a suitable term for the mine construction that holds the filler in confinement due regard was given other terms used in the Pennsylvania anthracite fields. The term "battery," probably the most widely employed, offered the objection of being a commercial term having electrical and mechanical significance. The term is also applied to wooden barriers for coal chutes in pitching beds. Its general use therefore might cause confusion.

The term "dam" is generally used to signify a water-tight permanent construction to confine large quantities of water within certain limits, the confining material being usually earth, stone, concrete, or

square timber. General usage in connection with hydraulic mine filling has developed this misnomer, and the term is frequently modified, as "slush," "flush," or "silt" dam.

The term "stopping" was rejected because of its general use in connection with the control of ventilation.

The term "bulkhead" is believed to be distinctive, and as used in this bulletin signifies a device having the usefulness of the type of retaining wall, constructed of various kinds of material, now being employed to confine filling material in the reclamation of beach-front lands.

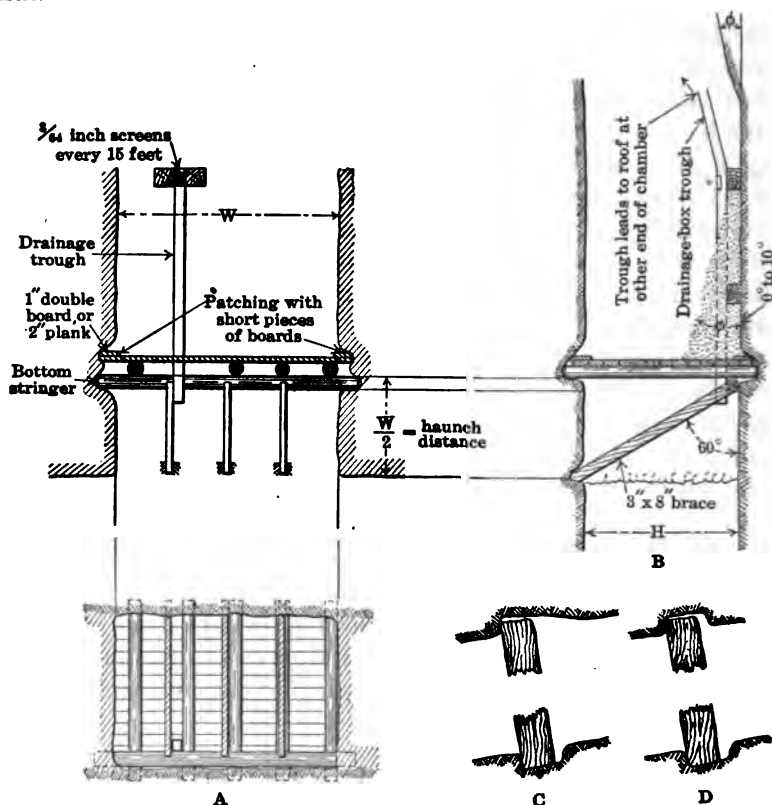


FIGURE 8.—Bulkhead and hitches for flat workings. A, plan view and front elevation; B, side elevation; C, prop held by a sliding and a trough hitch; D, prop held by trough hitches.

GENERAL RULES OF CONSTRUCTION.

The method of construction and the details of the various types of bulkheads used in common practice are greatly influenced by the geologic conditions, particularly by the inclination of the coal bed. The height of the bed, the width of the opening to be closed, and the character of the coal ribs into which hitches are cut for anchorage are also important factors.

as to make second mining possible at minimum expense.

The type of bulkhead used throughout the anthracite field is practically the same and is generally that first adopted, with the improvement of a few minor details. However, several details can be still further improved.

Mine workings (chambers) should be so opened as to entail the least possible expense in bulkhead construction. Instead of full-width chambers, the openings should be as narrow as haulage and ventilation will permit.

In general, bulkhead construction is considered a part of timbering. The haunch distance (fig. 8, A; fig. 9, and fig. 10, A)—that is, the distance along the rib from the entrance opening to the bulkhead hitches—should be equal to one-half the width of the opening for flat and chute workings and two-thirds the width of the opening for pitch workings.

As to the props used, their diameter in inches should be equal to their length in feet for flat workings, or to a multiple of corresponding lengths for chute and pitch workings, plus the depth of the top and bottom hitches in feet. The empirical formulas for the diameter of props are as follows:

$$\begin{array}{ll} \text{Flat workings} & \dots\dots\dots D = \frac{L}{2} \\ \text{Chute workings} & \dots\dots\dots D = \frac{3L}{2} \\ \text{Pitch workings} & \dots\dots\dots D = \frac{7L}{4} \end{array}$$

L=length of props, in feet.
D=diameter of props, in inches.

Formulas for the spacing of props are as follows:

$$\begin{array}{ll} \text{Flat workings} & \dots\dots\dots S = \frac{W}{H} \\ \text{Chute workings} & \dots\dots\dots S = \frac{2W}{3H} \\ \text{Pitch workings} & \dots\dots\dots S = \frac{4W}{7H} \end{array}$$

W=width of opening, in feet.
H=height of opening, in feet.
S=spacing of props, center to center, in feet.

Obviously the number of props required in a bulkhead is:

$$N = \frac{W}{S} + 1,$$

in which W represents the width of the opening in feet and S represents the spacing distance in feet.

The materials of construction to be employed depend upon the form and type of bulkhead to be erected. In general, it may be said

that props and plank, gob, dirt, blasted rock, concrete, and stone are usually adopted. In conjunction with these materials, limestone and calcareous shales have been used to neutralize sulphuric acid in the water. Packing, necessary to avoid excessive leakage and to form a filter curtain or "screener," may consist of burlap or brattice cloth, hay, shavings, manure, ashes, straw, forest leaves, etc.

Props and planks or boards are used for practically all flat workings. Timber, gob, stone, and concrete bulkheads are generally constructed where the dip of the coal bed is steep and the workings are comparatively new. The bulkheads can then be utilized to assist ventilation before actual filling is undertaken. Packing is used only on inclined workings and the quantity is almost proportional to the degree of inclination. Its purpose is to filter out of the seepage water any filler held in suspension. Care must be exercised not to make bulkheads too water-tight, as allowance must be made for partial relief of hydraulic head.

DETAILS OF BULKHEAD CONSTRUCTION.

Bulkhead construction, although chiefly in charge of the boss timberman, generally receives the careful attention of the colliery officials in charge, who possess thorough knowledge of the geological as well as the physical features of the workings to be filled.

After the section of the mine to be filled has been determined, the selection of bulkhead locations is considered. The materials suitable and the type of bulkhead to be constructed are carefully investigated, especial attention being given to the cost of the materials. The cutting of hitches, vertical and horizontal, is left to the foreman in charge, who is thoroughly familiar with the character of the coal and the rock at the determined locations. In general practice the depth of the hitches is equal to the diameter of the props. The vertical hitches are cut into the coal ribs parallel to the plane of the props; the horizontal hitches are cut into the bottom (floor) and the roof along the line of intersection of the plane of the props. This plane deflects from the normal^a up the pitch 2 to 10 degrees, depending on the dip of the bed. If the dip is less than 80 degrees, the deflection, as a rule, is equal to about one-seventh of the dip.

TIMBER BULKHEADS.

Timber can be used for all kinds of bulkhead construction, but is especially suited for use in flat and chute workings.

^a By normal plane is meant a plane at right angles to the plane of inclination, or dip, of the coal bed.

the depth of those in shale or "slate." The bottom hitch is generally of the trough type (fig. 8, D), whereas the top is more frequently of the sliding type (fig. 8, C), especially where the roof is hard. The spacing of props usually follows the formula previously cited for flat workings ($S = \frac{W}{H}$). For example, for a room 8 feet high

by 24 feet wide the props should be 8 inches in diameter ($D=L$) and spaced 3 feet apart center to center. By formula $N = \frac{W}{S} + 1$

a single row of props would be sufficient for a chamber of such size. If the roof and the floor are of shale or "slate," the hitches would be cut as trenches 12 inches wide and 8 inches deep. Where the roof and the floor are sandstone the hitches would be cut separately 10 or 12 inches square, but only one-half as deep as in the shale or "slate." This difference in cutting is made because shale or "slate" exposed to air and moisture swells or "heaves," causing in many places serious cracks in the bottom or roof and initiating so-called "water squeezes," whereas in sandstone such action seldom occurs.

The props are securely wedged at the top and manure or dirt is firmly packed around the bottom after the bottom plank has been attached, so as to avoid leakage. One and one-half inch plank or double 1-inch boards are then nailed to the inside face of the props for the entire height. The bottom, sides, and top are carefully joined or "patched" with short pieces of board nailed to the main boarding and the solid coal or pillar. If the dip of the bed is between 5° and 15° , a straw, hay, burlap, or brattice-cloth packing is used; or, in order to avoid heavy hydraulic head, a drainage trough to draw off excess water may be constructed as illustrated in figure 8, A and B.

BULKHEADS FOR CHUTE WORKINGS.

Chute workings, as previously defined, are understood to indicate workings in beds inclined more than 10° and less than 25° and where the lining of chutes with sheet iron is necessary to cause the coal to slide to the foot of the chamber on the loading platform. The heavier hydraulic pressure on bulkheads in such workings naturally demands a more massive and substantial construction than is required for flat workings. The character of the overlying and the underlying rock influences the construction to a certain degree as regards obviating water squeezes.

As an example, take a chamber 24 feet wide in a bed 10 feet high that dips 20°. The diameter of the props should be not less than 10 inches, preferably 15 inches, according to the formula $D = \frac{3L}{2}$. The hitches should be cut at least 10 inches deep in the bottom and not less than half that depth in the roof. The spacing of the props, according to the formula $S = \frac{2W}{3H}$, would be 19 inches from center to center. A single row of props would suffice if properly braced to either the roof or the bottom, depending upon the

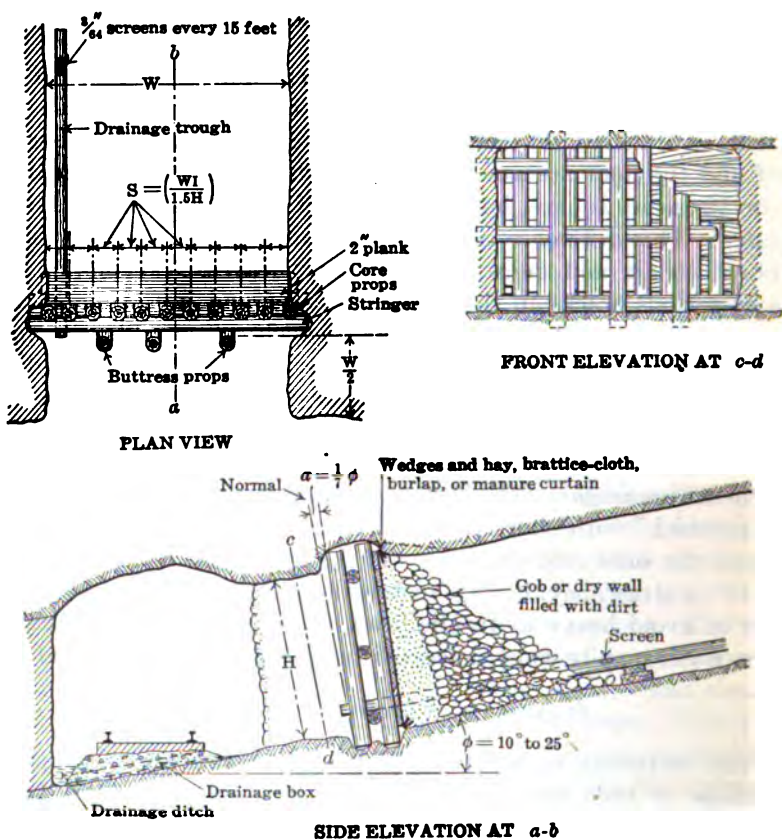


FIGURE 9.—Bulkhead for chute workings.

character of each. The method of construction is illustrated in figure 9. Drainage for this type of bulkhead is best accomplished by means of a trough, as illustrated in figure 9. This method avoids heavy hydrostatic pressure on the bulkheads and makes the water used available in the shortest time for reuse for hydraulicking, a feature that is of vital importance in nearly all mines where filling is done.

in many places a wall of dry gob is constructed to act as a partial filter.

BULKHEADS FOR PITCH WORKINGS.

Pitch workings, as previously defined, include workings in which the inclination of the bed exceeds 25° and in which the coal will run in chutes or troughs not lined with sheet iron. As the pressure on bulkheads in such workings is greater than on the bulkheads previously mentioned, it is obvious that the construction and the materials must be heavier and more substantial.

Assume a chamber 10 feet high by 24 feet wide. The diameter of the props, according to the formula $D = \frac{7L}{4}$, should be $17\frac{1}{4}$ inches or, say, 18 inches; the hitches should be of the trough type, from 10 to 18 inches deep, and wide enough to allow manure and dirt packing around the props and bottom plank. The spacing, according to the formula $S = \frac{4W}{7H}$, should be 1.4 feet, or, say, 17 inches, from center to center. As this spacing is less than the maximum diameter of the props, and as, by the formula $N = \frac{W}{S} + 1$, the number of props required is 17, two rows will be necessary. The extra props should be so distributed as to make the total supporting value equal to that of the entire number of props required, and they should be so placed as to give a clearance of at least 6 to 8 inches; therefore, substituting in the formula $N = \frac{W}{S} + 1$,

$$\frac{24}{1.5 + 0.5} + 1 = 13$$

The number of props to be placed in the first row is therefore 13; the remaining 4 ($17 - 13$) props are for reenforcement. The reenforcing props might be applied as three horizontal braces, as indicated in figure 9.

If the bulkheads are intended to withstand great pressure in beds of steep dip, the form of an inverted V is most preferred (fig. 10).

The bottom and top hitches are cut continuous, as are the rib hitches. When timber is used, the sticks are laid parallel to the dip of the bed in the form of a V, pointing up the dip, the sides of the V making angles of about 30° with the ribs. The timbers are carefully scored and hewed before being placed. In the bottom hitches a layer of fine manure and dirt is placed as a bedding for the first layer of timbers. The planking is placed vertically, a minimum num-

* 1.5 represents diameter of props, 18 inches.

† 0.5 represents 6-inch clearance.

ber of nails being used for fastening. Vertical buttress and horizontal timbers are placed as indicated in figure 10, A or B.

In this form of construction it is at times highly desirable to provide the "screener" and the gob-wall filter mentioned above.

Masonry, rather than timber, can be used for this form of construction. If masonry (fig. 11) is employed instead of timber, the form of the bulkhead is that of a full-struck arch, a semicircle with a radius

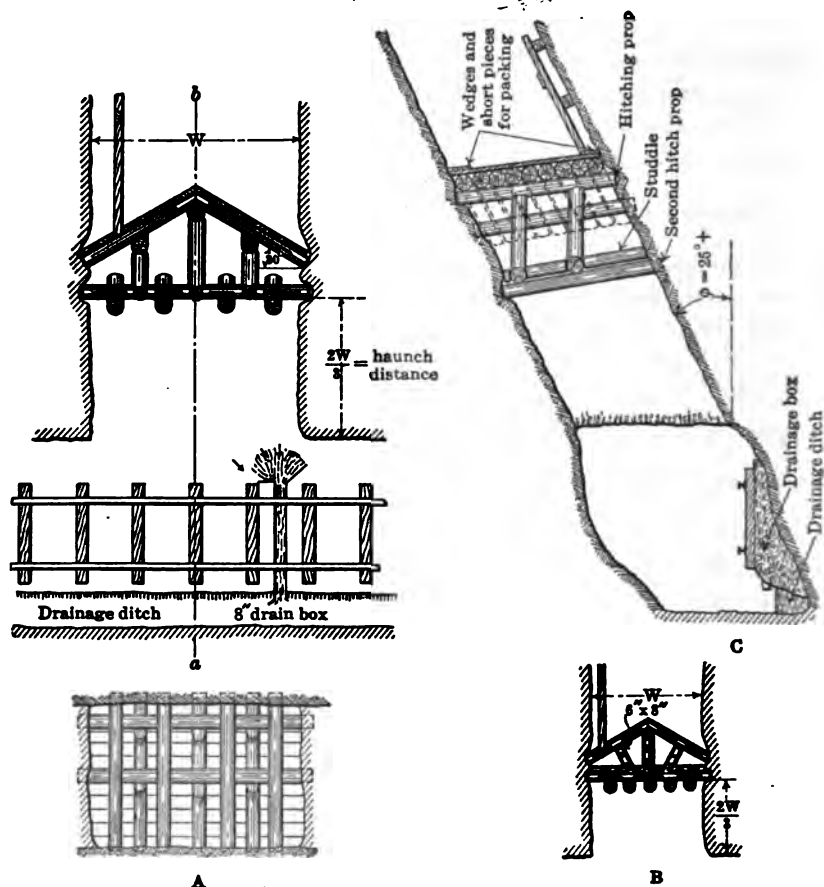


FIGURE 10.—Bulkhead for pitch workings. A, plan view and front elevation; B, plan view of modification of bulkhead shown in A; C, side elevation at ab.

equal to the width of the opening; the thickness of the bulkhead at the haunches is equal to one-third of the height and at the crown to one-sixth of the height. Drainage troughs (fig. 11) must be provided.

Under special conditions dry walls are in places used in conjunction with the timber construction. It is impractical to formulate any general rules for using such walls. However, it is safe to state that, wherever the inclination of the coal bed is over 18 degrees dry walls are invariably constructed if rock is available. Great faith is placed in this type of bulkhead. The specific purpose of the dry wall is to

the bulkhead of a portion of the direct hydrostatic pressure.

The use of rock^a blasted from the roof and floor for bulkhead construction was no doubt suggested by encountering the natural bulkheads formed by the many roof falls in old workings and the percentage of filling represented by the fallen rock. It may be possible to work out a practical application of blasted rock when the cost of other material for bulkheads becomes prohibitory.

Concrete bulkheads have been tried in several mines, and although their first cost is high they prove fairly economical in pitching beds

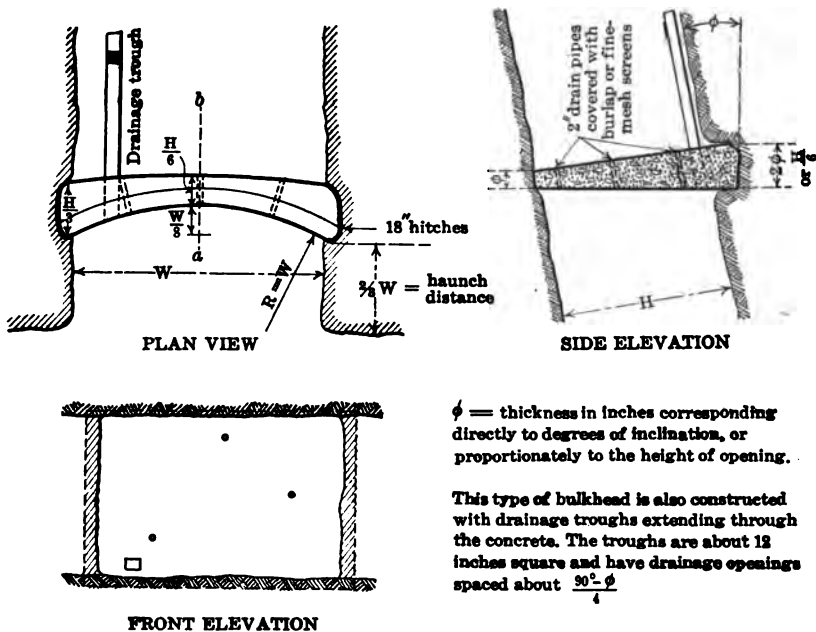


FIGURE 11.—Concrete or brick bulkhead for pitch workings.

where the bulkheads are subjected to great pressures and where the necessary timber would be very expensive.

This form of bulkhead construction is more frequently used where hydraulic filling is planned as a feature of the development of the mine. The bulkheads are constructed with trap doors so as to allow their use in conducting ventilation and in mine inspection in addition to their ultimate use as filler containers. The trap doors also permit traveling by workmen and officials before filling starts. After second mining has been started the bulkheads are of service as anchorages for new bulkheads for filling the opening on the mining side of a pillar.

^a Griffith, William, and Conner, E. T., Mining conditions under the city of Scranton, Pa.: Bull. 25, Bureau of Mines, 1912, p. 57.

With concrete bulkheads great care must be exercised to insure drainage by means of troughs.

Brick and stone bulkheads are seldom used. The remarks under concrete bulkheads are equally applicable to bulkheads of these types.

Concrete and other types of masonry construction possess unusual advantages when rehydraulicking is undertaken, especially in flat and to a large extent in chute workings. Second filling in any section of a mine is not undertaken until pillar mining, either partial or total, has been completed in that section. Timber bulkheads seldom last until second filling is begun, consequently the rib anchorage, either on one or both sides, is practically useless at the time of second filling, so that a quantity of valuable coal must be left as "anchor stumps" for anchorage for new bulkheads required in filling the openings made by mining the pillars. This objectionable feature is eliminated by masonry types, which allow a satisfactory connection or anchorage directly to the old bulkhead.

COST OF BULKHEADS.

PLANK-AND-TIMBER TYPE.

FLAT WORKINGS.

The cost of a given type of bulkhead depends largely on the amount of coal or roof that has fallen and the character of the coal and rock at the point of location.

Assume that a timber-and-plank type of bulkhead for filling a chamber 24 feet wide by 6 feet high is to be built in flat workings, with roof and floor of medium hard sandy shale. The itemized cost of construction is approximately as follows:

Itemized statement of cost of building a timber-and-plank bulkhead for flat workings.

MATERIAL.

Seven 6-inch props 7 feet long, at \$6.50 per ton-----	\$2. 00
350 feet 2-inch plank, at \$25 per M-----	8. 00
Nails and spikes -----	.35
Burlap or brattice-cloth drainage-hole coverings-----	.35
	\$10. 70

LABOR.

Cleaning and preparing bulkhead location, 2 men, one-half day-----	3. 00
Cutting hitches (top, bottom, and sides), 2 men, 1 day---	6. 00
Cutting, setting, and wedging props, 2 men, 2 days-----	12. 00
Planking and patching, 2 men, 1½ days-----	9. 00
Portage of material (approximately)-----	5. 00
	35. 00
Total-----	45. 70

For hard sandstone roof and bottom, other conditions being the same, there will be an additional labor cost for cutting hitches. This work necessitates moiling, and its cost will amount to at least three times that given above, or \$18. Therefore in sandstone, flat workings, the total cost of such bulkhead would be \$45.70 plus \$18, or \$63.70.

CHUTE WORKINGS.

In chute workings with medium hard, sandy-shale roof and bottom the cost of construction will be decidedly greater, as indicated below.

From the formula $S = \frac{2W}{3H}$, the props should be spaced 2 feet 9 inches

apart, center to center; therefore the bottom hitches would be of the box type and the top hitches of the continuous or sliding type. By

the formula $N = \frac{W}{S} + 1$, the number of props is 10. By the formula

$D = \frac{3L}{2}$, the diameter of the props is 9 inches. The itemized cost of the bulkhead would be approximately as follows:

Itemized statement of cost of building a timber-and-plank bulkhead for chute workings.

MATERIAL.

Ten props 9 inches in diameter by 7½ feet long (12-inch bottom hitch, 6-inch top hitch)-----	\$3. 00
350 feet plank-----	8. 00
Nails and spikes-----	. 35
Burlap or brattice cloth-----	♦ . 75
1,200 feet 1-inch boards for drainage troughs-----	30. 00
	————— \$42. 10

LABOR.

Cleaning and preparing bulkhead location-----	3. 00
Cutting hitches (top, bottom, and sides)-----	9. 00
Cutting, setting, and wedging props-----	18. 00
Planking and patching-----	9. 00
Portage of material-----	8. 00
Building 24-inch dry-wall filter-----	15. 00
Carpentry (drainage trough)-----	12. 00
	————— 74. 00
Total-----	116. 10

For hard sandstone roof and bottom, other conditions being the same, the cost of labor for cutting the hitches will be about three times that given above, or \$27; therefore the total cost of a bulkhead in chute workings with such roof and floor would be \$116.10 plus \$18, or \$134.10.

PITCH WORKINGS.

In pitch workings, the floor and roof being medium-hard sandy shale, the cost of construction would, of course, be higher than in flat or chute workings. By the formula $S = \frac{4W}{7H}$, the spacing of the props would be 2.3 feet. The props should, therefore, be spaced not over 26 inches apart, center to center. From the formula $N = \frac{W}{S} + 1$, the number of props required would be 12. The diameter of the props, determined from the formula $D = \frac{7L}{4}$, would be 10.5, or say 11 inches. The clearance distance between props ($S - D$) would be 15 inches; therefore to avoid expensive cutting in hard sandstone the bottom hitches should be of the box type and only the top hitches of the trough or sliding box type. The cost of the bulkhead would be approximately as follows:

Itemized statement of cost of building a timber-and-plank bulkhead for pitch workings.

MATERIAL.

Twelve props, 11 inches in diameter by 7½ feet long (12-inch bottom hitch, 6-inch top hitch).....	\$13. 00
350 feet 2-inch plank and patching.....	8. 00
Nails, spikes, burlap, etc.....	2. 00
Boards for drainage troughs.....	30. 00
	\$53. 00

LABOR.

Cleaning and preparing bulkhead location.....	5. 50
Cutting hitches (top, bottom, and sides).....	15. 00
Cutting, setting, and wedging props.....	23. 00
Planking and patching.....	12. 00
Constructing and placing drainage trough.....	12. 00
Building dry-wall filter and manure curtain.....	18. 00
Portage of material.....	20. 00
	105. 50
Total.....	158. 50

For hard sandstone roof and bottom the cost of labor for cutting the hitches, moiling being necessary, will be approximately three times that given above, or \$45; therefore the total cost of a bulkhead in pitch workings with sandstone roof and floor would be \$158.50 plus \$30, or \$188.50.

If the bulkhead being considered were of the inverted-V type, the cost of construction would be approximately as follows:

Itemized statement of cost of building a timber-and-plank bulkhead of the inverted-V type.

MATERIAL.

33 timbers, 12 inches in diameter by 16 feet long-----	\$52. 00
Other material same as itemized above-----	40. 00
	\$92. 00

LABOR.

Cleaning and preparing bulkhead location-----	6. 00
Cutting hitches-----	21. 00
Cutting, scoring, and placing timbers-----	66. 00
Planking, wedging, troughing, etc-----	42. 00
Portage of material-----	28. 00
	163. 00

Total-----	255. 00
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For hard sandstone roof and bottom an additional cost of \$42 should be added, making the total cost \$297.

CONCRETE TYPE.

The cost of a concrete bulkhead for use in filling the chamber referred to above (24 feet wide by 6 feet high) would be \$52 for material and \$173 for labor, a total of \$225, if the roof and the floor were of medium-hard sandy shale.

For hard sandstone roof and floor an additional cost of \$68 should be added, making a total of \$293.

COMPARISON OF VARIOUS TYPES OF BULKHEADS.

TIMBER BULKHEADS.

In comparing the advantages of the various types of bulkhead construction, the ease with which timber for timber bulkheads can be transported over caved and broken ground is an important factor favoring that type, as is the comparatively low cost of timber construction and its flexibility and advantageous application under difficult conditions. In timber construction, however, it is impossible to anticipate the reuse of materials employed.

MASONRY BULKHEADS.

Masonry construction is necessarily expensive. For extensive final or pillar mining the masonry type of bulkhead possesses attractive advantages and may in future work be developed so as to present

the greatest ultimate economy. Heretofore such constructions have been built primarily for ventilating purposes, being later used advantageously in mine filling.

BLASTED-ROCK BULKHEADS.

Bulkheads made by blasting the roof and floor are an extension of the plan of supporting the roof in this way.^a With a considerable quantity of small coal and fine dirt available it may be possible to construct a barrier of blasted rock across a comparatively flat working place and, by placing a sufficient quantity of this fine material at the inner face of the rock barrier, to form a filter that will eventually retain the filler. Judging from the apparent effectiveness of general roof falls as bulkheads, this type of construction should be efficacious and economical in sections considered inaccessible as regards the construction of either timber or masonry bulkheads, especially in places where the roof and bottom are softer than the usual sandstone of the anthracite mines and where the opening is less than 6 feet high. Air and water cause newly broken-down shales and "slates" to slack or disintegrate, consequently the voids or interstices in such a bulkhead rapidly fill up and the barrier becomes less pervious.

BRICK AND OTHER MASONRY BULKHEADS.

Brick, concrete, and other masonry bulkheads are the most expensive, but can be made to serve a double use, as ventilation stoppings and subsequently as filler bulkheads. In many mines where the workings are extensive it is necessary to construct stoppings of masonry to insure a sufficient quantity of air reaching the working faces. Bulkheads of masonry are most practical when filling is conducted on the panel or district system. In such practice each panel or district is separated by building bulkheads across openings in the boundaries only; no bulkheads are constructed at the foot of chambers or rooms. Timber bulkheads would be impracticable, owing to the long interval between the first mining and the time when filling can be safely undertaken.

FILLING DIFFERENT CLASSES OF WORKINGS.

As previously stated, hydraulic mine filling originated in connection with the solving of special problems under the stress of emergency. The initial efforts were naturally conducted along experimental and inefficient lines; improvements in method were developed as time for observation elapsed and results could be care-

^a * Griffith, William, and Conner, E. T., Mining conditions under the city of Scranton, Pa.: Bull. 25, Bureau of Mines, 1912, p. 57.

estimate where the filler was being deposited and whether it was accomplishing the desired results, rapid improvement has been made. In the past many large areas of mine workings were made entirely inaccessible by the filler blocking the only open places leading to the district to be filled. These effects led to attempts (1) to control the flow of the filler, and (2) to confine the filler within predetermined limits. Through gradual development the methods now extensively employed both in this country and abroad came into use. To-day hydraulic filling has wide application and is, under certain conditions, an important and indispensable aid in the mining of coal.

CLASSIFICATION OF SYSTEMS AS TO AREA TO BE FILLED.

If the area to be filled be taken as the basis of classification, all systems of hydraulic mine filling may be divided into three groups, as follows: Individual, panel, and collective.

In the individual system of filling the filler is discharged directly into each chamber or breast to be filled. This system is generally adopted in flat workings for either total or partial filling. In the panel system the filler is discharged into groups of chambers or breasts, one or more chambers or breasts being left unfilled. This system is generally adopted for partial filling in flat, chute, or pitch workings. In the collective system the filler is discharged at the highest accessible elevation in a given section of the mine and is allowed to flow unrestrictedly into all the chambers or breasts to be filled in that section. This system is adopted for total filling in chute or pitch workings.

CHOICE OF SYSTEM ACCORDING TO AREA TO BE FILLED.

The determination of the most desirable system of filling is dependent on the proposed future operation, the time to elapse between filling and pillar mining, and the ultimate pillar extraction and possible refilling. If the interval between filling and reopening is to be comparatively short, a relatively short-lived material may be used in constructing the bulkheads; for example, if reopening is to be undertaken within one year's time, oak props and hardwood plank will prove efficient and economical, and in refilling, the new bulkheads can be satisfactorily anchored to the old bulkheads. If the interval is to be longer, more durable wood—for instance, pine props and hemlock or pine planks—should be employed.

The method of reopening filled areas should be determined by the inclination of the workings. If the bed is flat, the gangway and airway are generally maintained; therefore the filling should be con-

ducted so as to require the least quantity of material to be rehandled, especially where only a partial recovery of pillars can be undertaken.

If the workings are inclined over 25° , the collective system is the most practical and economical, because, although bulkheads of heavy construction and consequent high cost will be required, comparatively few bulkheads will be necessary. Also, as the reopening of such workings consists in driving a road along the high rib of the old gangway, the least amount of pillar coal is isolated, and the maximum recovery is possible at the least expense in rehandling filler. In fact, with this system the immediate availability of a large quantity of coal somewhat compensates for the high expense incurred.

In flat workings, if the greater part or all of the pillar coal is ultimately to be extracted, the collective system possesses a great advantage over the individual or panel system, as the only bulkheads necessary are those provided for ventilation and for entrance and exit for haulage. However, in mines generating inflammable gas the panel system is the only one to be recommended, as this system allows free circulation of the ventilating current around the filled district at all times and thereby guards against dangerous accumulations of such gas.

Collective filling is the least expensive system, panel filling ranks next, and individual filling is the most expensive system.

In any form of filling, whether individual, panel, or collective, it is important to guard against the dangers of overflows or "rushes." These are frequently caused by piles of gob and dirt or by falls of roof. The water and the filler accumulate behind such obstructions until an overflow or break occurs. Then the onrushing mass often breaks bulkheads or blocks airways, haulage ways, and manways.

CLASSIFICATION OF SYSTEMS AS TO CHARACTER OF THE WORKINGS.

If the character of the workings be taken as a basis for classification, hydraulic mine-filling systems may be divided into four groups, namely: Surface openings, flat workings, chute workings, and pitch workings.

SURFACE FILLING.

The first group may be made to include all manner of filling conducted outside of the mines, whether for the purpose of grading or storing refuse. Strictly speaking, such filling is not usually done by a hydraulic process. The filler is dumped from wagons or cars in certain locations, either as spoil banks or for temporary storage.

The simplest form of surface filling consists in depositing material, either by the hydraulic method or in a dry state, in such cavities as are caused by mine caves or by "crop falls." All such filling is done

buildings.

In this connection it is well to bear in mind that a mine cave is caused by the removal or the robbing of pillars or the failure or crushing of pillars owing to a squeeze, and the resulting surface subsidence may be either uniform or intermittent.

Another use of surface filling, usually hydraulic, is for improving the topography at a colliery by filling small gulleys, depressions, washouts or even marsh lands. The advantage of such filling is readily appreciated by those who have had experience with colliery operations in rolling or swamp land.

Hydraulic filling has not yet become general throughout the entire anthracite region and unfortunately there still exist practices long since proven inconsistent with progress; these practices no doubt account for much of the highly regrettable pollution of the streams, sometimes unintentional and at other times willful. A remedy for such conditions should be eagerly sought and adopted. Unintentional stream pollution results from the erosion, by heavy rains, of culm temporarily or permanently placed upon steep watersheds, or the washing away of large spoil banks so placed as not to be beyond the limits of flood waters.* In many instances culm and other mine refuse have been placed in large piles at points close to breakers or washeries for the express purpose of temporary storage pending recovery and shipment of small sizes of coal. Within the last decade anthracite practice has been improved to meet a steadily widening market for the small sizes, millions of tons of which had for many years been stored in huge piles and had been considered as practically waste. In the reclamation of these piles, there is left as refuse only such culm as will pass through a screen with openings of three thirty-seconds of an inch, or in some instances of three sixty-fourths of an inch.

The fine refuse embraced in the coal loss is deposited on the surface or is hydraulicked into the mine workings. If it is deposited on the surface by the hydraulic method, the usual practice requires that a timber-and-dirt dam be constructed to contain it, or that the depressed area be inclosed by a bank of rock or breaker refuse. When the material has been deposited the height of the reservoir walls can be increased 3 or 4 feet by banking up the deposited material. The water draining from the filler may be collected at some convenient point and re-used in the breaker or washery or allowed to run off.

* Report of Pennsylvania commission appointed to investigate the waste of coal mining with the view to the utilization of the waste, 1893, p. 134.

FILLING FLAT WORKINGS.

Hydraulic mine filling in flat workings, or workings of less than 10 degrees inclination, is the most difficult to perform, and is the most expensive. Filling in such workings must be done almost entirely by the use of pipes; therefore great care must be exercised to provide a supply of water at all times sufficient to transport the material through the pipe lines. The official in charge should possess some knowledge of the working principles of hydraulics. The proper size of the transport pipes should receive special study. For example, if the amount of material to be hydraulicked is less than 200 cubic yards per day of 9 hours, and if the cost of handling water is comparatively high, it may be possible to economize on the quantity of water and to conduct satisfactory filling with 4-inch pipe, especially if the pipe line is required to conform to an irregular profile and if the mean effective hydraulic head is high enough to keep the pipe line clear. For lower mean effective hydraulic head, for a larger quantity of material, and for varying grades obviously a pipe of larger diameter, say 6 or 8 inches, must be used.

To avoid blockages, it is essential to maintain the pipe line so that the filler will flow through at the greatest possible velocity. A good practice is to allow the water to flow through the pipe line for at least 10 minutes before the filler is introduced and for a sufficient time at the close of a day's operations to insure all filler being out of the line.

WATER REQUIRED.

It is obvious that the quantity of water required is dependent on the mean effective hydraulic head; the proportion of water varies from 50 to 90 per cent, depending on the grade of the line and the nature of the filling material. Naturally the steeper the grade the less water is required. In general practice the proportion of water approximates 90 per cent of the total volume passing through the line.*

DISTRIBUTION OF FILLER.

One of the most essential conditions in hydraulic mine filling, especially in filling flat workings, is absolute control of the flow until the filler reaches the desired point of deposit. Good practice requires that the pipe line be laid to this point, no allowance being made for flow in the chambers. This method will guard against the dangers of "rushes," with a consequent possible loss of output or life if the filling is being conducted in workings on a "live" lift

* Wilson, H. M., Irrigation engineering, revised edition, 1910, pp. 61-69, 338-344.

accomplished by constructing branch pipe lines into different sections of the mine workings previously prepared to receive the filling material, so that it will be possible to divert the flow from one section to another at stated intervals or upon short notice. The practice of continuous filling in one section is to be discouraged, as the best results are obtained only by allowing from 15 to 18 hours to elapse after 200 to 400 cubic yards has been deposited, so that proper seepage can take place before another quantity is deposited.

To allow for refilling or supplementary filling, a period of two weeks should elapse before the filler pipe line is withdrawn from any section considered filled by the first filling. The filler will shrink from 1 to 10 per cent during the seeping period. (Pl. I.)

In all horizontal places or places of low dip overflow or "telltale" pipes should be so placed as to indicate by discharge when the filler deposited has reached the roof at the highest point in those places.

LABOR REQUIRED.

The working force necessary satisfactorily to operate the filler line is, roughly speaking, one man for the surface, one man to patrol each 1,000-foot length of operating pipe line, and one man to inspect pipe discharge, filler deposit, and seepage. The underground force can also look after the drainage watercourses leading to the sump.

VENTILATION.

The proper ventilation of mine workings while being filled requires more attention than is usually accorded that feature. Ventilation is especially important in mines where explosive gases are generated. If filling is conducted on the individual system, means for allowing the escape of air from behind the bulkheads should be provided, thus preventing dangerous accumulations of explosive gases. Means should also be provided for carrying away the heat and moisture usually given off during the shrinkage of filler consisting of anthracite culm. Where excessive amounts of dangerous gases are evolved it is advisable to conduct filling on the panel system so as to permit circulation of air around the entire area to be filled. If none of the above precautions is practicable it may be possible to extend the drainage troughs in such a manner that they can be utilized for ventilation purposes. Direct connection may be made to the return air course or to a split air current either on the return or the intake side. A "booster" fan driven by steam, compressed air, or electricity has been found very serviceable.

SEEPAGE.

The drainage of the seepage water from the filling material deposited requires careful attention so that the least possible amount of filler shall be carried away in suspension. The very fine material, called sludge, that escapes from the deposited mass generally consists of the particles essential to perfect cementation. Great care should be exercised to retain the sludge in the body of the mass, instead of allowing it to seep out and cause troublesome accumulations in the main sumps. Proper seepage will not take place if the filler is deposited continuously or at frequent intervals, but it is obtained to a satisfactory degree if filler is deposited only during periods not exceeding four hours.

GENERAL REMARKS.

As previously stated, the practice of filling flat workings by continuous flow should be discouraged for several reasons. The most important of these are: (1) The filling is incomplete both horizontally and vertically; (2) the workings can not be properly ventilated while filling is in progress; and (3) improper seepage takes place. These objections are eliminated to a great extent by using either the individual or the panel system previously described.

There is opportunity for general improvement of the present system of preparing mine workings for filling. The change most essential is in the manner of opening chambers and of driving crosscuts. New chambers should be opened as narrow as possible, allowing only sufficient room for ventilation and necessary car clearance. The opening width of the room neck might be reduced, for a distance equal to the width of the opening, to about two-thirds that of the regulation width of chamber, thereby decreasing the cost of bulkhead construction by at least one-third and providing better conditions for reopening.

FILLING CHUTE WORKINGS.

Chute workings—that is, workings where the inclination ranges between 10° and 25° —as a rule are free from gas accumulations except along the axis of an anticline; therefore, the danger from gas is practically negligible and the panel system may be discarded for either the individual or the collective system, depending on the degree of efficiency desired. The comparative advantage in cost lies in the smaller number of bulkheads required and the great saving in pipes or troughs.



A. GANGWAY CUT THROUGH CULM FLUSHING, OXFORD MINE.



B. GANGWAY CUT THROUGH FLUSHING, CULM STANDING AS A VERTICAL SELF-SUSTAINING WALL, OXFORD MINE.

WATER REQUIRED.

Considerably less water is required in chute workings than in flat workings to carry the filler to the point of deposit. In many mines a proportion of water as low as 60 per cent is sufficient if the profile is not too irregular and the mean effective hydraulic head is comparatively high.

DISTRIBUTION OF FILLER.

In the distribution of filler the usual practice is to construct a pipe line along the airway of the gangway or lift next higher than the section to be filled (fig. 12, lines A, B, C, D, and E), or if there is an impregnable chain pillar between the chamber faces and the next higher lift or level, the pipe line is led through the headings or cross-cuts at the face of the chamber of the filling lift. If the section to be filled consists of more than five chambers, T connections with valves are placed at intervals of five chambers, for instance, at the third, eighth, thirteenth, etc. (fig. 12, line A), the pipe line being made long enough to supply at least three groups of five chambers each. When filling is started and the number of chambers ready to receive the filler is greater than 15 the filler may be conducted from the various connections for half-shift periods. In the case of the workings shown in figure 12 the extreme inside section (chamber 13) and the adjoining section (chamber 8) should preferably receive filler the first day. The following day filler should be placed in the extreme outside (chamber 3) and the second adjoining section (chamber 13), and so on. This shifting allows each section to drain for 20 or more hours before filler is again deposited in it. The filler may be discharged from the T connection at the face of the chamber and a ditch or channel through the loose dirt may be dug by shovel all the way to the bulkheads, thus saving the cost of pipe or trough. Under such conditions it is possible in the individual or panel systems to fill fairly satisfactorily five chambers from the bulkheads to the point of discharge. Complete filling, or refilling after seepage from first filling has stopped, can be done, after all chambers have been filled as previously outlined, by having the pipe line discharge at the highest point in the entrance to that section (five chambers), bearing in mind the necessity of recovering all the pipe in that section.

If the collective system is employed and another section is not available for filling, the pipe line should be so constructed as to permit the discharge of filler at intervals of 10 chambers, the first point of discharge being at the fifth chamber in, the second point of discharge at the fifteenth chamber in, and so on.

The filler shrinks considerably during the seeping period, and to permit refilling or supplementary filling a period of two weeks

should elapse before the filler pipe line is withdrawn from any section considered filled by the first filling.

In figure 12 the figures along the pipe lines A, B, etc., refer to the mining periods, the figures 1, 1, 1, etc., referring to the first mining, 2, 2, 2 to second, etc. The arrows indicate the direction of the mining.

LABOR REQUIRED.

The working force for operating the filler line for chute workings generally consists of one man on the surface, one man to patrol each 1,500 feet of operating line, and one man to direct the pipe discharge, the chamber flow, the deposit of filler, and the seepage. The underground force also looks after the drainage water courses.

VENTILATION.

If pillar mining is to follow the filling of chambers, it is advisable to utilize the inside or line chamber and the airways to the gangway as air courses. This can be done by constructing bulkheads in the crosscuts between the two chambers farthest in and between the gangway and the airway. The general practice is to maintain the gangway and the airway as intakes and the inside or line chamber as the return-air course of all lifts. The volume of air passed should be sufficient to carry off gas and keep the temperature from rising too high.

SEEPAGE.

Careful attention should be given to the character of the seepage. This will depend on the construction of the bulkheads and the system of depositing the filler. It should be borne in mind that the fine particles often carried away in suspension in the run-off water are very necessary for the most efficient cementation of the filler and should be retained therein. Rivulets or freely flowing streams should be guarded against, as they are the surest source of breaks or leaks in bulkheads. The ends of drainage troughs should be provided with strainers made of brattice cloth or canvas.

GENERAL REMARKS.

The general remarks under the section entitled "Filling Flat Workings" apply equally to the filling of chute workings, except that in chute workings filling by unrestricted flow is decidedly more efficient than in flat workings, but the practicability of its adoption under any given conditions should be established by careful study.



The hydraulic filling of workings in which the inclination exceeds 25° is less difficult than that of flat or chute workings and the method of depositing the material is much less expensive. Fewer bulkheads need be constructed and less attention to chamber flow is required. The most satisfactory system of mine filling is that which allows inspection from the point of introduction to the point of discharge. This is possible where shafts or slopes form the connection between the mine workings and the surface. When the filler is introduced through crop falls or cave holes and is allowed to flow by gravity it will be unconfined and neither the places where it is being deposited nor the solidity of the filling will be known. The practice should be discouraged.

WATER REQUIRED.

When the workings are comparatively open the quantity of water required varies from 40 to 90 per cent of the total volume.

DISTRIBUTION OF FILLER.

The distribution of the filler may be accomplished by means of pipes and pressure in flat parts of the workings and by means of troughs in inclined places. The pipes may be of wood, wrought iron, cast iron, or steel, with various modifications. The troughs may consist of ordinary boards or planks nailed together, usually in the shape of a V, or of half-round terra-cotta or cast-iron sections. Sheet-iron plates bent square, hexagonal, V shape, or half round have been successfully employed. The use of troughs is neither popular nor economical unless the grade is rather steep—that is, greater than 10° —and a large quantity of water is available.

The general practice is to conduct all filler through metal or wooden pipe to the point of discharge in the chambers, the distribution being generally the same as that previously outlined for chute workings. The filler may be conducted through mine workings in wood, metal, or earthen troughs if there is no ascending grade in the line. Filler has been conducted along old drainage ditches and through chambers to desired sections with considerable success. Usually only such bulkheads are constructed as are necessary to confine the district and to provide proper openings for ventilation and drainage for other parts of the mine as well as for the district being filled.

The collective and panel systems are used almost exclusively. The choice between these systems is dependent on the physical condition of the workings. Mine filling as a rule is not undertaken until a

comparatively long time has elapsed after the first mining, and consequently many large areas of workings may be practically inaccessible on account of falls, caves, or squeezes. Under such conditions it is necessary to introduce the filler in such manner as will insure the most efficient distribution and filling. It is therefore advisable to have several points of discharge in a district, and thereby increase the possibilities of completely filling around the falls and caves. For filling under such conditions considerably larger quantities of water are required than are necessary under ordinary conditions.

If possible, filling should start in the lowest lifts and progress toward the higher workings. Where final filling, or filling after the pillars have been mined, is conducted, the final step in filling a given section of the mine should be the filling of the air course.

LABOR REQUIRED.

The working force necessary to operate a filler line under the difficulties of pitch workings should be approximately as follows: One man to attend to the surface arrangement, one patrolman for every 1,000 feet of line (two men, if troughs are used), at least two men to direct the flow of the filler in the chambers, and one man to look after the drainage water flowing from the bulkheads to the sump. The man looking after the drainage may be placed in charge of the filling operations.

VENTILATION.

In pitch workings where inflammable gases are generated, and where pillar mining is to follow the filling, it is a safe practice to maintain one of the chambers as an air course despite the general tendency of the inflammable gases to accumulate at the higher elevations. The air course may be established by constructing substantial bulkheads in crosscuts and other openings connecting with the permanent air courses.

SEEPAGE.

The question of seepage under conditions encountered in pitch workings is of the greatest importance and demands careful study and consideration. The first consideration should be the thorough draining off of water after the filler has been deposited. The most satisfactory method is by means of draining troughs with strainers, as previously described. If proper seepage is thus insured, excessive pressures due to the hydrostatic head of the plastic mass, which endanger the stability of the bulkheads and may endanger the lives of the workmen in lower levels, are avoided. Proper seepage, including the prevention of loss of fine particles, will insure perfect cementation of the filler, thereby insuring the greatest supporting value. Such cementation is especially important in order that the

The drainage water from the filler should be conducted along abandoned haulage ways, or airways, or through old chambers, to the permanent sump. If this is impracticable, the drainage water may possibly be collected at some convenient point outside of the place of deposit, and be conducted in pipes down the pitch, and if necessary, a considerable distance upgrade to the regular or permanent sump. This will avoid serious erosion along traveling ways and haulage ways, and guard against possible large accumulations of water which might be a source of considerable damage.

GENERAL REMARKS.

The general remarks under the section entitled "Filling Chute Workings" are applicable to "pitch" workings. A special caution should be observed, however, in the adoption of unrestricted filler flow. This is particularly dangerous when the filler has been deposited in a chamber from the bulkhead in the first crosscut to the adjoining vacant chamber. In the crosscuts, as a rule, temporary wooden stoppings are constructed for the control of the ventilation current; frequently the crosscuts are partly blocked with "chipping" coal or with small falls of roof. The stoppings or blockages may cause the filler to accumulate and the hydrostatic pressure, as soon as it becomes great enough, may cause the mass to break through, rush down the vacant chamber, and strike the bulkhead with great force, disastrously affecting the operation of the mine and even causing loss of life among workmen in the lower levels.

Sedimentation of the filler in pitch workings is practically perfect. The filler is usually deposited so compactly as to make refilling or supplementary filling practically unnecessary until pillar mining is undertaken.

EFFECT OF HYDRAULIC MINE FILLING ON VENTILATION AND DRAINAGE.

The ventilation and the drainage of a mine are directly influenced by the filling. The entire ventilation is improved by the general filling of old workings. In unfilled workings air stoppings, if made of wood, rot and become leaky, the constant partial combustion of carbonaceous matter evolves considerable carbon dioxide gas, and the coal itself takes up oxygen. Such conditions are highly objectionable. Furthermore, through the filling of chambers and entries the frictional resistance to the air current may be greatly diminished, thus allowing, with the same water gage, a much larger quantity of air to be circulated through the mine; provided, of

course, that the main upcast or downcast is of larger cross-sectional area than the main mine air course. The ventilating pressure necessary is considerably decreased by the filling of the "dip" workings.

The drainage of a mine is often seriously affected by the introduction of mine filling. Water courses must be changed so as to obtain steeper grades, or the water must be piped, because the increased proportion of solids carried in suspension causes frequent accumulations that threaten the flooding of haulage roads or pumping stations. Special provision must be made to intercept and to remove as much sediment from the water as is possible before it enters the sump (fig. 13, detail sections). In most instances the introduction of hydraulic mine filling requires additional pumping facilities. If the water contains acids, special wood or cement lined pipes and pump parts are required. On the other hand, when an inadequate pumping plant is replaced by one of the requisite capacity, the total cost of handling the mine water is often materially reduced.

PREVENTION OF ACCUMULATIONS OF GAS AND DUST.

Next to the reclamation of coal in pillars probably the greatest advantage that can be claimed in favor of mine filling is the resulting prevention of dangerous accumulations of gas and dust in the dry old workings of a mine where such accumulations may seriously jeopardize its safe operation.

Accumulations of inflammable gas in whatever quantity are dreaded by all miners. It is serious enough when such accumulations occur in readily accessible places, but to have them occur in old workings, practically inaccessible on account of falls and caves, is a cause of deep concern to every mine worker and official. By careful management serious accumulations have been removed by displacement with filler in mines where removal by ventilation would have entailed enormous expense in clearing caves and in reconstructing air courses. The removal of such accumulations is not an easy matter under favorable circumstances, especially when the gas mixture has been stagnant for some time and partial stratification has taken place. Numerous explosions of gas have resulted from accumulations in unsuspected places as a result of falls of roof or caves blocking the air currents or causing permanent stoppings to leak, and to short circuit the air current, thereby causing insufficient air to reach the working places or the faces of the air courses.

Accumulations of fine inflammable coal dust are to be guarded against as dangerous. Mine filling serves to reduce to a minimum the dust-collecting surfaces in old workings. Accumulations of dust are liable to become active propagating agencies in the event of an explosion, so that any practicable method of eliminating this danger is desirable.

perature of the surrounding atmosphere has been noted, particularly where iron pyrite was present in the filler. Although such generation of heat at first caused considerable alarm as possibly indicating a fire, experience has demonstrated that it has a beneficial result. Owing to the rise in temperature from the heat developed by the iron pyrite or other oxidizable material, the humidity of the air is increased. The moisture taken up by the air is condensed in cooler parts of the mine, and helps to keep the dust there from becoming dry.

Practically the only gases caused by mine filling are carbon dioxide (black damp), carbon monoxide (white damp), and hydrogen sulphide (stink damp). In most mines the quantities are so small that they can not be detected by chemical analysis, and in general they may be said to be negligible.

DRAINAGE SYSTEM REQUIRED.

When hydraulic mine filling is practiced, a well-planned and well-equipped drainage system is essential. Not infrequently the amount of mine filling is restricted by the quantity of water the existing pumping equipment will handle. On a sound business basis, however, additional facilities should be provided, even though the cost may amount to thousands of dollars.

Drainage is considered under three headings, as follows: (1) Watercourse to sump; (2) pumping equipment; and (3) sump cleaning.

WATERCOURSE TO SUMP.

The course along which the water draining from the filler must flow on its way to the main or an auxiliary sump should receive careful attention. The water should be so conducted as to offer the least hindrance or possibility of danger to the operation of the mine. Where there is a sufficient continuous grade, the water may be confined to ordinary mine-dirt or mine-rock channels. Where the grade is less, wood or terra-cotta troughs may be employed to advantage, particularly where blockage of the ditch or channel may cause serious accumulations of water and fine filler, which, when suddenly released, cause serious washouts or floods on traveling-ways or haulage-ways. To guard against such danger, wood pipe or secondhand metal column pipe may be used to conduct the water beyond points of probable blockage. Pipe is also used to conduct water across basins, "swamps," or other depressions.

fig. 13). By sedimentation a considerable part of the fine material

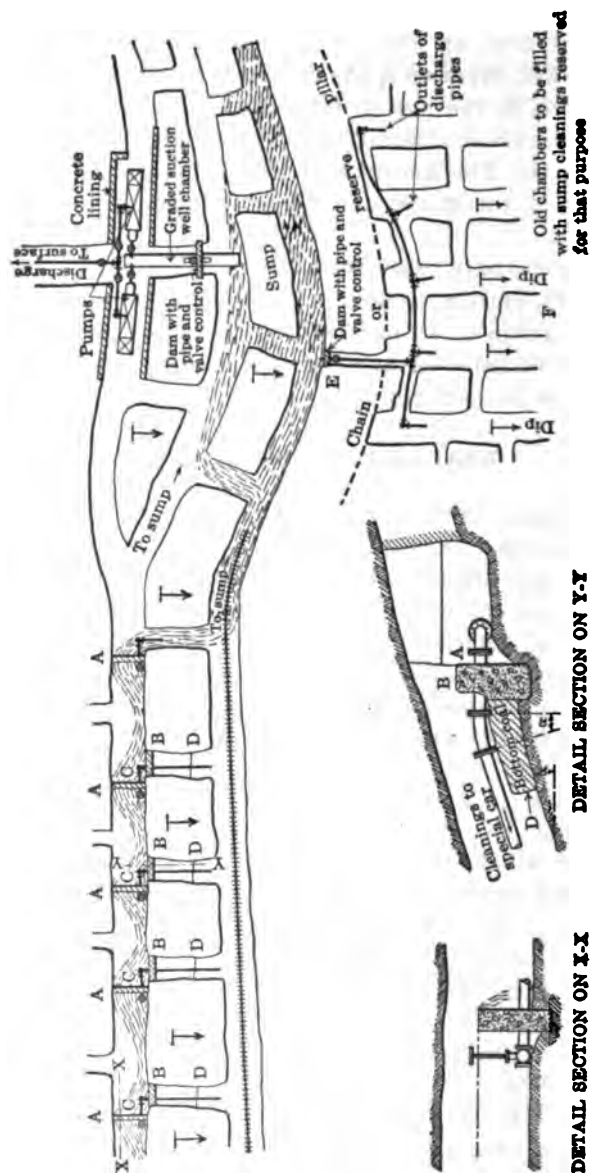


FIGURE 12.—Mine-drainage system.

still held in suspension by the water is eliminated. At intervals of inactivity the basins may be cleaned and the sludge loaded directly into cars, flushed into water-tight cars, or flushed into the mine workings on lower levels. The dams of the settling basins may be of

plank, stone, brick, or concrete. They usually are constructed so as to fill one-half of the opening along the lowest rib. This height allows cheap and convenient cleaning.

PUMPING EQUIPMENT.

If the lift to the surface is less than 500 or 600 feet from the main pump and the mine opening is not a shaft, the water-handling equipment generally consists of duplicate or triplicate units, either electrically driven or high-duty, triple-expansion, steam or compressed-air pumps located at the central drainage sump, generally near the foot of the shaft, slope, or water-discharge bore hole. There are also in use many steam-operated and a few electrically driven water hoists. The water ends of pumps and the tanks of water hoists are constructed of various acid-resisting metals. Pump valves and water barrels are also lined with cement and wood. Where comparatively little sediment is present in the sump water, the regular plunger type of pump is used. Where the water contains considerable sediment the piston type of pump is preferable.

SUMP CLEANING.

Sump cleaning always presents a serious problem and is generally expensive. Where the main central sump is located above the lowest working levels, cleaning is greatly simplified by driving a narrow opening through the chain pillar and constructing a substantial masonry dam in the opening (E, fig. 13). A 10-inch pipe with valve and blank flange should extend through the dam in the middle near the bottom. A pipe line may be laid to connect this pipe to vacant openings (F, fig. 13). When the sump needs cleaning, the accumulation of sludge is stirred by means of shovels or hoes, the blank flange is removed, the pipe line is connected, the valve is opened, and the entire sediment is easily and quickly flushed out. If the sump is so located as to make impracticable the method above outlined, a wooden track for mine cars is constructed to the lowest accessible point in the sump, and after the sump has been drained as much as possible, men shovel the sediment into cars, haul it to other parts of the mine, and unload it into old workings.

COST OF HYDRAULIC MINE FILLING.

The conditions under which hydraulic mine filling is conducted are extremely variable, and statements of cost must necessarily be prefaced with the remark that the geological and physical characteristics of a mine greatly influence and may determine the ultimate cost of filling its workings.

Folds and faults may make the slope of the filler pipe lines less than the hydraulic gradient, thus preventing filling by gravity and necessitating the pumping of the material. As a rule, the physical condition of old or practically abandoned mine workings considerably increases the cost of transporting materials and of constructing bulkheads and pipe lines. In the summaries presented below, the estimates given are based on the present cost of labor and materials in the anthracite region.

SURFACE TRANSPORTATION.

Hydraulic mine filling with culm commences at the breaker or washery, and therefore the detailed discussion of cost logically begins at that point.

Surface transportation may be divided into two general classes: (1) Gravity and (2) mechanical. In one the filler and water flow by gravity to the beginning of the intermediate system, in the other the filler and water are conveyed from some common point at the breaker or washery to the intermediate-transportation system by either pumps, elevators, scraper or conveyer lines, or cars.

The cost of surface transportation depends largely on local conditions. If the filler is only such refuse as results from screening and washing coal and will flow by gravity to the intermediate-transportation system, the cost will vary from 1 to 1½ cents per cubic yard. If to the screenings and washery refuse, boiler ashes and crushed breaker rock are added, to the cost given must be added the cost of crushing, which, including running expenses and depreciation charge, varies from 3 to 4 cents per cubic yard, making the total cost of gravity transportation 4 to 5½ cents per cubic yard.

If the refuse is conveyed by a conveyer or scraper line to dump cars and hauled in cars, an additional charge of 3½ to 4½ cents per cubic yard must be added to the above. If instead of being handled by the scraper or conveyer and the dump car, the filler is handled by pump through pipe lines not exceeding 500 feet in length, the total cost of transporting the filler to the intermediate-transportation system should not be greater than 2½ to 4 cents per cubic yard.

The cost of transporting on the surface local sand, gravel, loam, and clay, none of which requires crushing, should not exceed 14 cents per cubic yard. The cost of transporting in large quantities local quarried material that can be handled by steam shovel and sluiced by gravity from the crusher to the intermediate-transportation system will be 30 to 40 cents per cubic yard. Such quarried material must be crushed so that all particles will pass through a ¾-inch screen. If the crusher is at a considerable distance from the intermediate-transportation system, a charge of 0.4 cent per cubic yard for every 500

The total cost in cents, C, may be expressed by the formula:

$$C = M + \left(\frac{D}{500} \times .4 \right),$$

in which M represents the cost in cents of the material as it comes from the crusher, and D represents the distance in feet from the crusher to the intermediate-transportation system.

The utilization of distant material, such as sand, gravel, loam, or crushed rock and slag, that might be brought by empty coal cars returning from points outside of the coal fields and convenient to the railroads, has been a much-discussed problem among able mine managers and mining engineers. The cost of the material at the source, the cost of loading and unloading, and the freight charges constitute a prohibitory expense under existing conditions. The freight charge of 1 cent per ton per mile, added to the cost of the material and the labor, makes the use of distant material rather a remote practicality unless some plan of transportation can be evolved that will be of benefit to both the operator and the transportation company.

It must be borne in mind that most of the railroads traversing the anthracite region depend upon the coal trade to furnish from 20 to 70 per cent of their total freight tonnage, and perhaps it may yet be possible for them to devise favorable freight rates for filling material. It is recognized that the problem of freight rates is far-reaching and must be approached advisedly; however, until some equitable method of dividing the expense can be worked out, mine filling involving long transportation of the filler will not be popular.

COST OF INTERMEDIATE TRANSPORTATION.

Intermediate transportation commences at the point where surface transportation ends. The crudest and cheapest mode of intermediate transportation is introducing the filler, without pipe, through crop falls, cave holes, or abandoned shafts. Practically the only cost connected with this method is that of preventing the blocking of openings with filler or of preventing the caving of the side walls.

A more efficient, and consequently more expensive, method is to introduce the filler by means of pipes, either in shafts or slopes. This entails the cost of a pipe line and its maintenance. Generally speaking, the lines consist of wood-lined cast-iron pipe, with flange or bell-and-spigot connections, or wood-stave pipe; the cost of handling filler with either kind of pipe is about the same, varying from one-fourth to one-half cent per cubic yard of filler passed through. Each type offers desirable advantages; the metal pipe possesses greater

strength to withstand shocks by roof falls or runaway cars, and the wood pipe possesses the advantage of being in shorter and lighter lengths. Metal pipe is seriously attacked by acid water, whereas wood pipe is practically immune from such attack.

Bore-hole transportation is generally employed where the mouths of slopes or shafts or other mine openings are above the drainage level; that is, the elevation of the railroad tracks at the loading chutes. It is by far the least expensive method, the cost varying from one-tenth to two-tenths of a cent per cubic yard.

COST OF UNDERGROUND TRANSPORTATION.

Underground transportation may for convenience be divided into three classes: (1) That in which the filler is allowed to flow unconfined; (2) that in which troughs are used; and (3) that in which pipes are used.

The cost of unconfined flow is practically negligible. The cost of trough transportation varies from one-tenth of a cent per cubic yard for wood troughs to three-tenths of a cent for metal troughs. The cost of pipe transportation per unit of 500 feet varies approximately as follows:

Kind of pipe.	Cost per cubic yard, cents.
Steel -----	0.30 to 0.35
Wood -----	0.35 to 0.40
Cast-iron -----	0.35 to 0.40
Wrought-iron -----	0.40 to 0.45

The physical and chemical properties of the various types of pipe, together with the conditions to be met, must determine which type will be the most economical for installation in a given mine.

COST OF DISTRIBUTION.

DISTRIBUTION ON THE SURFACE.

Distributing the filler on the surface entails the cost of conveying the material to a point where it will flow by gravity, of controlling the flow, and of depositing the filler at or near the bulkheads or retaining walls. The cost of handling, as stated under the heading "Surface Transportation," varies from 4 to 7 cents per cubic yard; to this must be added 1 to $1\frac{1}{2}$ cents per cubic yard for flow control, and from five-tenths to seven-tenths of a cent per cubic yard for bulkhead construction; this aggregates a cost for surface distribution varying from $5\frac{1}{2}$ to 9 cents per cubic yard.

DISTRIBUTION IN FLAT WORKINGS.

The cost of distribution in flat workings includes attendance and bulkhead construction. The cost of attendance varies from 1 to $1\frac{1}{2}$ cents per cubic yard of filler and the cost of bulkhead construction from $6\frac{1}{4}$ to $7\frac{1}{2}$ cents per cubic yard.

1½ cents per cubic yard for attendance and from 11 to 12½ cents per cubic yard for constructing bulkheads for individual filling. The cost when the panel or the collective system is used is considerably less.

DISTRIBUTION IN PITCH WORKINGS.

The cost of distributing filler in pitch workings varies from 1 to 1½ cents per cubic yard for attendance and from 13 to 26 cents per cubic yard for bulkheads, depending on the type of bulkheads constructed and the system of filling employed. The cost for the panel or the collective system is considerably less than for the individual system.

COST OF DRAINAGE AND VENTILATION.

The cost of drainage and ventilation in connection with hydraulic mine filling is difficult to determine on account of the great variety of possible contingencies. The item of controlling the drainage water from the bulkhead to the sump is a simple matter in so far as maintaining the drainage pipes, troughs, or ditches, and constructing and cleaning the settling basins is concerned, the cost varying from one-fourth to one-half a cent per cubic yard handled. The cost of handling the water used for transporting the filler varies widely; if the usual pumping plant can easily handle the additional water, the cost is comparatively negligible, varying from 0.1 to 0.15 cent per cubic yard, but if additional pumping equipment, such as relay pumps, is required, the cost may vary from 1½ to 8 cents per cubic yard of filler deposited. Under favorable geological conditions the drainage from several adjoining mines may be collected into a common sump and all the water lifted to the surface by high-duty pumps or by water hoists at considerably less cost than the figures quoted.

The cost of rearranging ventilation, if rearrangement is necessary, is practically negligible; if there is any appreciable cost it is overbalanced by the increased volume of air going into the working places and by the lessened frictional resistance so that the item may legitimately be charged to ventilation in the regular operating expense of the mine.

RECAPITULATION OF COSTS.

A recapitulation of the costs of hydraulic mine filling is given in the table following.

For surface filling, the cost of depositing breaker refuse, culm, or ashes, varies from 5 to 9 cents per cubic yard.

*Recapitulation of cost of hydraulic mine filling.**

Material.	Surface transportation.			Intermediate transportation.		Underground transportation.		Distribution.			Drainage.		Total.	
	Gravity.	Mechanical means.		Shaft or slope.	Bore hole.	Trough.	Pipe.	In flat workings.	In chute workings.	In pitch workings.	Incidental.	Special.	Minimum.	Maximum.
		Scraper or conveyor.	Pump.											
Culm.....	1-14 4-54	44-6 74-10	24-4 54-8	1-14 4-14	1-14 4-14									
Culm mined with crushed breaker and boiler refuse.														
Local hydraulicked sand, loam, gravel, clay, etc.	8-14	114-124	94-104	1-14	1-14			74-9	124-144	14-274	1-14	11-8	94	544
Local crushed sand, loam, gravel, clay, etc.	20-40			1-14	1-14									
Material brought from a distance in returning empty coal cars.	The cost of the material, loading, freight, unloading, preparing and surface transportation.			1-14	1-14	25 per cent should be added to each cost specified above.								

* The figures in this table represent cents per cubic yard and are based on data obtained from operating plants with a capacity of at least 400 cubic yards daily.

CONCLUSION.

In concluding this preliminary report it seems fitting to refer to the lack of publicity so far given in this country to the subject of hydraulic mine filling. Although it has engaged the attention of mining men for many years, comparatively little has been published in this country concerning practical results. The contrary is true in Europe, especially as regards the application of the process in Upper Silesia. Engineers from that Province visited this country in 1893, gathered considerable information, and introduced the practice in Germany. Many articles regarding application, cost, and advantages of the process have appeared in foreign periodicals.

It is hoped that the investigations conducted by the city of Scranton^a and by the Pennsylvania State Anthracite Mine Cave Commission, appointed by Gov. Tener in 1911, and the scientific inquiries of the Bureau of Mines will be productive of results that will offer relief to anxious owners of the surface overlying mine workings, and to owners of industrial plants located in districts where subsidence may occur. These investigations should also prove of much service in aiding to lessen waste in the mining of valuable minerals other than anthracite.

^a Griffith, William, and Conner, E. T., Mining Conditions under the City of Scranton, Pa., report and maps: Bull. 25, Bureau of Mines, 1912, 89 pp., 29 pls.

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- DAVIES, J. B.** A novel method of conveying culm into old workings to support the roof. Col. Eng., vol. 14, August, 1893, p. 11. Describes method of introducing culm and water at the Black Diamond colliery, near Kingston, Pa.
- Pillar drawing. Mines and Minerals, vol. 20, February, 1900, pp. 289-291. Illustrated description of the method used for taking out pillars when the workings have been flushed with culm.
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- ENGINEERING NEWS.** Pumping coal to market. Vol. 31, February 22, 1894, pp. 159-160. Discusses the feasibility of the scheme as indicated by an exhibit at the World's Fair in Chicago in 1893. The coal was reduced to an impalpable dust, mixed with an equal weight of water, and thus conveyed through pipes.
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PUBLICATIONS ON MINE ACCIDENTS AND METHODS OF MINING.

The following Bureau of Mines publications may be obtained free by applying to the Director, Bureau of Mines, Washington, D. C.:

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